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LEAD AND ZINC MINING AND MILLING IN THE UNITED STATES

CURRENT PRACTICES AND COSTS

(ABSTRACT)

FROM AN UNPUBLISHED BULLETIN MANUSCRIPT

BY

CHAS. F. JACKSON, JOHN B. KNAEBEL, AND C. A. WRIGHT

(ABSTRACTED BY CHAS. F. JACKSON)

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LEAD AND ZINC MINING AND MILLING IN THE UNITED STATES
CURRENT PRACTICES AND COSTS[1]

(Abstract)

(From an unpublished bulletin manuscript)

By Chas. F. Jackson,[2] John B. Knaebel,[3] and C. A. Wright[4]
(Abstracted by Chas. F. Jackson)

INTRODUCTION

This circular is an abstract of a manuscript completed in August 1932, which cannot be published in its entirety due to lack of printing funds. It was originally planned to publish the manuscript as one of a series of bulletins dealing with the mining and milling of ores of the principal metals.

Because of the common association of lead and zinc minerals in many ore deposits, and because a considerable proportion of the zinc produced in this country is obtained as a by-product of lead mining or vice versa, it seemed necessary to treat the mining and milling of lead and zinc ores in the same volume.

Object and Scope of Paper

This paper was written to combine in a single volume a discussion of the modes of occurrence of lead and zinc ores and the methods and costs of mining and milling these ores, with descriptions of typical mining and milling practices in the United States.

Production

Statistics on lead[5] and zinc[6] production of the world and of the United States have been compiled for 1801 to 1927, inclusive, and show the trend in production rates and changes in the distribution of production here and abroad. Total domestic production of refined lead varied but little during 1925, 1926, and 1927 and averaged 688,815 short tons per year. Domestic production of primary zinc during the same period averaged 594,628 short tons per year.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U.S. Bureau of Mines Information Circular 6776."

2 Principal mining engineer, U.S. Bureau of Mines.

3 Assistant mining engineer, U.S. Bureau of Mines.

4 One of the consulting engineers, U.S. Bureau of Mines.

5 Smith, Lewis A., Summarized Data of Lead Production: Econ. Paper 5, Bureau of Mines, 1929, 44 pp.

6 Pehrson, Elmer W., Summarized Data of Zinc Production: Econ. Paper 2, Bureau of Mines, 1929, 47 pp.

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The present paper deals principally with methods and costs of mining and milling lead and zinc ores, and in this connection the tonnage of ore mined and milled is more pertinent than the amount of metal produced. In 1929, 161 lead and zinc mines in the United States having individual outputs valued at $100,000 or more produced 21,298,748 tons of ore yielding an average of 91.3 pounds of combined lead and zinc per ton. In 1930, 19,360,710 tons of lead and zinc ore were won from 160 mines producing more than 1,000 tons each.

From the data available it was not possible always to determine definitely whether lead or zinc predominated at a given mine; in a few instances, the precious metals may have contributed greater returns than the lead and zinc. Table 1 gives the production of crude ore by regions in 1930, the classification as between lead, zinc, and lead-zinc ores being based upon the most reliable information available as to the predominating metal at each property included in the summarized figures.

TABLE 1.- Crude ore mined, in 1930 by regions, tons

Region	Lead ore	Zinc ore	Lead-zinc ore	Total
Rocky Mountain States, California, and Washington	1,166,451	339,754	2,567,336	4,073,541
Central States	6,735,102	1,307,125	5,101,317	13,143,544
Eastern States	.....	907,263	1,236,362	2,143,625
Totals	7,901,553	2,554,142	8,905,015	19,360,710

PART I

GEOLOGY OF LEAD AND ZINC DEPOSITS

Methods and costs of mining and milling are influenced by the geology and mineralogy of the ore deposits. For the purposes of this paper the discussion of geology of lead and zinc deposits is limited to those features which affect mining and milling problems.

Types of Deposits

The principal sources of lead and zinc ore in the United States are in deposits of the following types:

1. Deposits in flat beds of limestone, dolomite, or chert; typified by the deposits of the Tri-State and Southeastern Missouri districts. The greatest production in the United States, in terms of tons of crude ore, comes from deposits of this type.

2. Tabular deposits, in the form of fissure veins or occupying fault or shear zones, dipping at various angles, typified by the deposits of the Coeur d' Alene district of Idaho, some of the Utah deposits, and others in the Rocky Mountain States.

3. Replacement deposits, usually in carbonate rocks and genetically related to igneous activity. The ores of the Leadville, Tintic, Park City, and Edwards (N.Y.) districts are well-known examples of this type.

4. Deposits due to igneous metamorphism, usually remote from contacts. The best known examples of this type in the United States are the deposits of Magdalena, N. Mex., and the Franklin and Sterling Hill ore bodies in New Jersey.

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Mineralogy of Zinc and Lead Deposits

The commonest and most important minerals of lead and zinc are the sulphides, galena and sphalerite, and the bulk of the production is derived from sulphide ores. The carbonates, however, are important ores in some districts.

In a few districts the mineralization is simple; thus, in the Southeastern Missouri district the lead ores are in general composed of galena in a nonmetallic gangue, with almost no silver and little zinc except locally. On the other hand, the deposits of Mascot, Tenn., and of Edwards, New York, are mined for zinc and contain little lead.

More often, however, the lead and zinc minerals are closely associated in primary ores. Silver is present in most of the western ores, and many of them also contain varying amounts of copper, iron, and gold, and sometimes vanadium, molybdenum, nickel, cobalt, arsenic and antimony. Some of these ores are worked profitably only because they contain appreciable amounts of silver.

Character of Outcrops

The character of the outcrops of the ore deposits is determined largely by the relative susceptibility of the ore and the enclosing rocks to weathering and erosion, the climate and topography, the shape, size, and dip of the deposit, and the visual appearance of the minerals in the ore after they have been altered by oxidation and weathering. If the ore contains a quartz gangue and occurs in limestone or some other easily eroded rock the outcrop is apt to stand out prominently. If the primary ore contains much pyrite, in addition to quartz, the outcrop often will consist of a conspicuous mass of honeycombed quartz stained with yellow or brown limonite.

In districts like those of Missouri and Wisconsin, where the ore occurs in flat beds, there may be no outcrops whatever, or they may be inconspicuous. Zinc sulphide alters readily to sulphate and relatively speaking is easily dissolved by surface waters, and although in some deposits (such as those of Leadville, Colo.; Tintic, Utah; and Cerro Gordo, Calif.) important oxidized zinc ores occurred at surface, zinc ore is more often confined to the sulphide zone. Galena is one of the most inert of the common sulphides and frequently is found at or near the surface. In some deposits where lead predominates near the surface zinc is found to predominate at depth. The bulletin manuscript discusses the changes in lead and zinc deposits near the surface, in the oxidized zone, and below water level.

Localization of Ore Shoots

In many districts it has been found that the ore shoots are largely confined to certain favorable beds and are low grade or lacking in others; when such a phenomenon has been clearly established, it assists greatly in the prospecting, development, and mining of the ore. Structural features, such as faults, breccia and shear zones, contacts, intersections of fissures with other fissures or with certain formations, and folds, often influence the formation of ore shoots.

MINING DISTRICTS

Southeastern Missouri

The ores occur in flat-lying sedimentary rocks, chiefly in the Bonneterre limestone, which is 160 to 200 feet thick. Galena is the principal ore mineral, although in a few

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places there is enough sphalerite to make it profitable to produce both lead and zinc concentrates. The galena occurs principally disseminated through the limestone and shaly beds in horizontal sheets along bedding planes. Some of the ore bodies are 700 or 800 feet wide by over 1,200 feet long and range from a few feet to over 200 in thickness. The ores are low grade and yield on the average about 3.5 percent metallic lead, though locally they may be much richer.

Tri-State District

The deposits are found in the Boone formation, believed to have been originally a limestone but now made up of flat-lying beds of limestone, dolomite, chert, nodule beds, and one or more oolite beds. The deposits are related to structural features and occur in zones of shearing, shattering, and brecciation. They lie at shallow depths; the bottom seldom is more than 350 feet below the surface. From 1917 to 1929, inclusive, 6,504,369 tons of zinc concentrates and 1,221,823 tons of lead concentrates were obtained from 117,401,237 tons of mine rock. The average indicated recovery was 3.32 percent metallic zinc and 0.624 percent metallic lead, and the average metallic zinc content of the crude ore was about 4.32 percent.

Eastern Tennessee District

The ore occurs in dolomite beds which dip at about 18 to 22°. The ore mineral is sphalerite, which occurs in large part as veinlets and seams with secondary dolomite. The ore bodies are large in area and commonly range from a few feet to 50 or more in thickness and in places are 150 feet thick. The ore is low grade, and in 1929 the mine output averaged 2.90 percent zinc. The sphalerite is of exceptional purity, and practically no pyrite or other metallic mineral is associated with it.

Coeur d'Alene District

The rocks of the district consist of pre-Cambrian sediments, largely quartzites, shales, and some calcareous beds into which monzonite and syenite have been intruded in the form of irregular masses, dikes, and sheets. The formations have been intensely folded and faulted. The principal ore bodies are fissure veins or lodes formed largely by replacement of the brecciated and sheared country rock in fissures and sheeted zones in quartzite. Mineralogically, the ores consist of argentiferous galena, sphalerite, and pyrite with chalcopyrite in small but variable amounts, in a gangue of siderite and quartz. The ore shoots are characteristically persistent to great depths, and many are very large. The main ore zone at the Morning mine is 1,500 to 2,000 feet long, is practically continuous for 5,000 feet below the outcrop, and is 6 to 30 feet wide. The Bunker Hill ore bodies occur mainly as wide, irregular masses within a sheared area 1,000 feet wide by 6,000 feet long, dipping at 40 to 50°. At most of the mines the grade of the ore varies between the following limits: Lead, 3.5 to 11 percent; zinc, 1 to 12.5 percent; silver, 1.0 to 20.0 ounces per ton.

Park City, Utah

The ores are found chiefly in the Wasatch limestone, Weber quartzite, Park City limestone, and the limestones, sandstones and shales of the Thaynes formation. These sediments have been intruded by diorite and diorite porphyry, to which the ores are genetically related. The rocks have been uplifted and subjected to considerable folding and faulting.

The ore bodies are of two general types. The fissure type, found principally in quartzite, is characterized by widths of 3 to 80 feet and dips of 40 to 55°; and the bedded-

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replacement type, which provides 95 percent of the ore at the Silver King Coalition mines. Some of the Silver King ore bodies have been followed on the strike for more than 10,000 feet without notable change in size or grade. The principal ore minerals at Park City are galena, pyrite, blende, and tetrahedrite in a gangue of quartz, chert, calcite, fluorite, rhodochrosite, and country rock.

Tintic, Utah

At the Tintic Standard mine the major ore bodies are in limestone at or near the contact with faulted faces of quartzite. Individual ore bodies adjacent to the fault faces dip from 45 to 90°, and those distant from the fault faces are tabular replacements of flat-lying limestone beds. The tabular ore bodies are 6 to 200 feet high. The principal revenue from the mine is from lead-silver ores with a quartz-barite gangue. Occurring with lead-silver ore in large masses is a siliceous ore containing about 4.85 percent lead, 0.36 percent copper, 17.24 ounces silver, 0.037 ounce gold, 66.96 percent silica, and 11.50 percent iron. The lead ore shipments have averaged 25.04 percent lead, 0.31 percent copper, 30.29 ounces silver, 0.039 ounce gold, 49.03 percent silica, and 7.18 percent iron.

Butte, Montana

Copper, with some silver and a little gold, is found in the central part of the Butte district. The country rock is a quartz-monzonite variant of the Boulder batholith, cut by granite porphyry and aplite dikes. Bordering the central zone is a belt of more complex ore in which silver increases and sphalerite is found. Below the leached gossan there is usually a zone of silver enrichment, and below the water level the zinc content increases until, at depth, mines originally worked for silver have become primarily zinc producers. The principal minerals of the deposits, which occur in the form of veins, are quartz, pyrite, rhodonite, sphalerite, galena, rhodochrosite, and argentite. The veins of the Butte district have been intricately faulted and offset in at least two directions and are characterized by heavy gouge on the walls and in bands within the veins.

At the Black Rock mine, the principal vein dips at an average of 80° and ranges from a few feet to 120 feet in width. The country rock is granite, which has been intensely altered near the vein.

Barker District, Montana

The most important ore bodies occur in a fissure vein in syenite. The vein is crescent-shaped in plan and has been exposed in underground workings for 4,000 feet laterally and 1,200 feet in depth. The ore is banded and usually is 4 to 7 feet wide. Individual ore bodies are 400 to 750 feet long. Vein walls usually are quite firm but in places are blocky. The ore minerals are galena, marmatite, sphalerite, and pyrite, with considerable silver, in a gangue of altered syenite and rhyolite, calcite, quartz, barite, rhodochrosite, and marcasite. During 1929 the average assay of the ore milled was: Lead, 6.21 percent; zinc, 5.01 percent; silver, 9.09 ounces per ton. With increasing depth, the percentage of zinc has increased.

Pecos District, New Mexico

The ore bodies occur in a shear zone a few feet to several hundred in width, in which the igneous rocks have been altered to parallel bands of schist. The ore bodies replace the

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schist, and the mineralization is a mixture of sphalerite, galena, chalcopyrite, and pyrite containing gold and silver. The ore bodies have been developed over a length of 2,000 feet and down to the 1,200-foot level. They consist of irregular, disconnected lenses of sulphides, which pinch out abruptly both horizontally and vertically. The ore and walls are both generally very loose and require close timbering. During the period 1927, 1928, and 1929 the ore from the Pecos mine averaged 16.06 percent zinc, 3.73 percent lead, 1.02 percent copper, 3.39 ounces silver, and 0.109 ounce gold.

Santa Rita, New Mexico

The principal rocks consist of flows of rhyolite, andesite, and basalt, underlain by limestone, shales, and sandstones. The strata have been considerably faulted. Several types of deposits occur, including fissure veins, limestone replacements, disseminated copper ores, and contact metamorphic deposits, but the replacement type has produced most of the lead and zinc ore. Many of the deposits were first worked for precious metals in the zones of oxidation and enrichment; but the base metals, especially zinc, have become more important with depth. Conspicuous gossans marked the outcrops of many ore bodies. During 1929 and 1930 the ore milled at the Black Hawk concentrator averaged as follows: 2.0 ounces silver, 2.5 percent lead, 0.5 percent copper, and 12.0 percent zinc.

Yellow Pine District, Nevada

A thick mass of folded sedimentary rocks, over half of which is carbonate rocks, is exposed in the district. These rocks later were covered by igneous flows and tuffs, and still later sills and large masses of granite porphyry intruded the sediments in the vicinity of large thrust faults. Most of the ore is found in brecciated, dolomitized limestone. Many of the deposits are grouped about intrusive masses of porphyry, but others are several miles from any known igneous outcrops. Most of the zinc and lead deposits occur in lower Mississippian beds and, except for the Yellow Pine and Prairie Flower ore bodies which underlie a porphyry sill, are not near outcrops of intrusives. Lead and zinc usually occur together, although a few deposits carrying zinc are devoid of lead and vice versa. Most of the important ore bodies are of the flat or bedded type and are localized in breccia zones, and the ore occurs filling fractures and to a lesser extent replacing dolomitized limestone.

Edwards, New York

The zinc ore bodies in this district occur in a belt of crystalline dolomites, quartzites, schists, and gneisses, which are intruded by granitic magmas and pegmatites. These rocks have a general dip of about 45°, but local folding has contorted them into a hook-shaped plunging structure. Ore deposition was controlled by plunging minor folds, and the ore bodies are lenticular in horizontal section. They are 5 to 25 feet thick and 100 to 200 feet long along the strike and are persistent to depths of 2,500 feet down the dip. The ore is a mixture of marmatite and pyrite, but sometimes pyrrhotite takes the place of pyrite. Walls are not usually well-defined, and parallel streaks and bunches of ore sometimes branch off from the main ore bodies. The latter lie wholly within the dolomite; frequently they do not outcrop, and at the Edwards mine five of the seven known ore bodies failed to do so. They all dip with the formation (at 40 to 45°).

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Franklin Furnace District, N. J.

The deposits occur in crystalline limestone in a belt up to half a mile wide. Igneous Rocks now metamorphosed to gneiss, intruded the limestone along the western side of the belt. Most geologists believe the ore deposits are of contact metamorphic origin. The deposits are hook-shaped in plan and section, present the form of plunging troughs which pitch downward from the nose of the hooklike outcrop at an angle of 25 or 30° near the surface, and flatten to 10° or less at depth. The ore bodies are 10 to 100 feet thick and extend to more than 1,100 feet below the surface. They are valuable for their zinc, manganese, and iron, which occur chiefly as oxides and silicates. Typical ore consists of an aggregate of 50 percent franklinite, 20 to 30 percent willemite, 2 to 6 percent zincite, and 3 to 11 percent calcite.

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PART 2

EXPLORATION, DEVELOPMENT, AND MINING

Exploration

In this paper the term "exploration" is applied to the search for new deposits and the extensions of known ore bodies. During the past 4 years economic conditions have not encouraged the search for new ore bodies. On the other hand, some companies which are well-fortified with funds probably have employed this period to increase their ore reserves by methodical exploration of extensions of known ore bodies and search for ore bodies adjacent thereto.

Exploration may be conducted from the surface, underground workings, or both, depending upon the size, shape, and dip of the deposits, their depth below the surface, their position relative to surface topography and to accessible mine workings, and the relative costs of surface and underground work.

In the Tri-State district there are no surface indications of ore over most of the area where the largest productive mines are situated. They were located by widening the field of exploration from the earlier discoveries of surface ores in other parts of the district. In the Southeastern Missouri district the vast deposits of disseminated-lead ores were found by drilling in areas where surface deposits of an entirely different nature were first discovered. Similarly, the sulphide zinc ores at Mascot, Tenn., were explored by drilling in areas where oxidized surface ores had been worked. In the tri-State district the churn drill is used almost exclusively, whereas in southeastern Missouri the diamond drill is employed. These local preferences are due to the nature of the formations in each field rather than to cost or other considerations. At Mascot, both churn and diamond drills have been used.

Following are some actual churn-drilling costs from operations in the Tri-State district.

Property	Acres	Holes drilled	Holes in ore	Ave. depth of holes, feet	Cost of drilling
1	80	129	35	250	$35,000
2	104	129	52	279	53,980
3	320	162	49	326	66,015
4	105	120	43	305	37,482
5	160	219	84	330	79,497
6	60	90	.....	300	27,256
7	80	138	55	235	40,839
8	40	78	47	300	26,450

Often inclined tabular ore bodies outcrop, and they are explored by following them down the dip from the surface and laterally by underground methods. Other ore bodies do not apex at or even near the surface and underground methods, often supplemented by diamond drilling or test-hole drilling with hammer drills, are employed to locate and explore them. In both underground and surface exploration advantage is taken of all information gained previously in the district concerning the habit of ore occurrence, the influence of structural relationships, and favorable beds or other formations. Underground methods are prevalent in the Western States and at many mines are the only ones employed, while at others they are supplemented by diamond drilling or test-hole drilling with hammer drills.

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Netzeband[7] gives the following costs for test-hole drilling in the Tri-State district:

	Cost per foot
Labor, operating	$0.82
Labor, maintenance	.06
Repairs	.09
Explosives	.01
Power, oil, and depreciation	.24
Total	$2.12

According to Dobbel,[8] the total cost of deep-hole drilling with hammer drills at the Chief Consolidated Mines, Utah, was $0.97 per foot.

Summary of Exploration Practice

Exploration from the surface by churn or diamond drilling is the accepted method for exploring flat-lying bedded deposits, and in some important districts drilling is the only method employed before development and the installation of mining and milling plants. The choice between churn and diamond drills for exploring such deposits depends largely upon the nature of the ground, its hardness, homogeneity and whether badly fractured or not, which in turn affect the cost of drilling and the amount of core recovery possible with the diamond drill.

In prospecting tabular deposits of high dip the diamond drill is superior to the churn drill, since it is desirable to cut the ore at right angles to its dip, which requires inclined holes that cannot be drilled with a churn drill. Such deposits can often be best explored principally by ordinary underground methods. However, there are few deposits, even when assays of drill samples are unreliable, where diamond or test-hole drilling cannot be used advantageously to determine important structural relationships and to probe for blind lodes and isolated lenses of ore.

SAMPLING AND ESTIMATION OF THE ORE DEPOSITS

This subject has been covered in an earlier bulletin[9] and is discussed at some length in the bulletin manuscript of which the present paper is an abstract. A review of the sampling and ore estimating practices at numerous lead and zinc mines reveals that although drill sampling and channel sampling usually are employed in exploration and development work, these methods are not used as extensively for stoping control as at most gold, copper, and iron mines. Instead, visual inspection of working faces is relied upon to a large extent in many lead and zinc mines to control the grade of ore mined. Table 2 summarizes sampling and estimating practice at a number of lead and zinc mines.

7 Netzeband, W. F., Underground Deep-hole Prospecting at the Eagle-Picher Mines: Trans. Am. Inst. Min. and Met. Eng., vol. 75, 1927, pp. 35-41.

8 Dobbel, Chas. A., Deep-Hole Prospecting at Chief Consolidated Mines: Trans. Am. Inst. Min. and Met. Eng., vol. 72, 1925, pp. 677-689.

9 Jackson, Chas. F., and Knaebel, John B., The Sampling and Estimation of Ore Deposits: Bull. 356, Bureau of Mines, 1932, 155 pp.

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TABLE 2.- Summary of sampling and estimating practice

Mine or district	Method of sampling						Estimates of ore reserves
	Churn drills	Diamond drills	Hammer drills	Face samples: Pick or chip	Face samples: Channel or groove	Grab samples	
Southeastern Missouri		(1)	(2)				From diamond-drill samples.
Tri-State district	(3)		(2)				From churn-drill samples.
Mascot, Tenn.	(1)	(1)	(2)			(4)	From sample assays and estimated grades of working faces.
Edwards, N.Y.		(1)	(5)		(6)		From grades of adjacent stopes and from samples.
Page mine, Idaho					(7)		From sampling of development openings.
Hecla and Star mines, Idaho			(8)		(7)		Do.
Morning mine, Idaho			(8)		(7)		Do.
Silver King Coalition mines, Park City, Utah					(9)	(10)	
Park-Utah mine, Park City, Utah						(11)	Grade calculated from that of adjoining stopes.
Tintic Standard mine, Tintic, Utah					(12)	(11)	From groove-sample assays posted on assay maps.
Block P mine, Mont.					(13)		Not made.
Black Rock mine, Mont.					(14)		From sample assays
Pecos mine, N.Mex.		(15)	(8)		(16)		From cross-sections and all available assays.
Ground Hog mine, N.Mex.					(17)		From channel-sample widths and assays.
Gilman, Colo.			(18)				In part from composite samples of drill holes.

[1]Core or sludge assayed. [2]For testing backs, floors, and walls. [3]Cuttings assayed. [4]From development headings in ore. [5]Cuttings from deep-hole drilling assayed, results 25 percent high; short test-hole samples not assayed. [6]Not done regularly, assays 25 percent high. [7]Samples assayed for lead and zinc; assays for silver on composite samples. [8]For testing walls for parallel stringers and ore bodies. [9]In milling-grade ore. [10]5 to 10 percent higher in grade than mill samples. [11]Taken from cars at loading chutes. [12]From drifts, raises, and square-set faces in stopes. [13]From drifts and raises. [14]All development headings and stope faces. [15]Sludge assayed; core assayed if mineralized. [16]All underground openings showing mineralization. [17]From all drifts and crosscuts at 5-foot intervals. [18]Sampled in 6-foot lengths.

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MINE DEVELOPMENT

In this paper the term "development" is applied to the operations involved in preparing a mine for ore extraction and includes tunneling, shaft sinking, crosscutting, and drifting.

Existing exploration openings often may be employed as permanent openings, but it is frequently advisable to provide new ones, differently located, of a different type, or larger and straighter for more efficient operation of the mine. The bulletin manuscript discusses such subjects as: Shaft v. tunnel development; vertical v. inclined shafts; size of shafts; shaft-sinking costs; level development, level intervals, size of drifts and crosscuts, cost of drifting and crosscutting, and plan of levels; stope development; and cost of mine development.

Tables 3, 4, 5, and 6, present, respectively, data on typical shaft-sinking costs, drifting, and crosscutting, tons of ore produced per foot of development work, and current mine-development costs per ton of ore mined at lead and zinc mines.

TABLE 3.- Typical shaft-sinking costs

Mine and location	Shaft section, feet	Kind of rock	Depth sunk, feet	Cost per foot	Remarks
Leadand zinc mines					
No. 2 mine, Tri-State district 1	5 by 7; 1 compartment	Surface, shale, dolomite and flint beds	260	$20.08	Total cost to company.
Hartley-Grantham mine, Tri-State district 2	6 by 6	Do.	200	$7 in shale $10.50 in hard rock	Contract labor and explosives only.
				$18.00	Total cost to company.
Pim shaft No. 8 mine, 3 S.E. Missouri district	6 by 18 in clear; 4 compartments	-	730	$76.17 sinking only $92.56 sinking and concreting	Shaft partly concrete-lined.
Pecos mine, N. Mex. 4	6.67 by 20.33 outside of timbers	Metamorphosed igneous rocks	1,050	$119.01	Cost per foot of last 328 feet.
Other mines					
Ajax mine, Cripple Creek 5	6 by 15; 4 compartments	Granite	(1,481 to 1,983) 502	$57.28	Labor, $25.77; materials, $16.93; service charges, $14.58.
Sylvanite, Ontario 6	6 by 16; 3 compartments	Syenite porphyry	963	$71.09	Sunk from 1,000 level.

1 Netzeband, Wm. F., Method and Cost of Mining Zinc and lead at No. 2 Mine, Tri-State District, Picher, Okla.: Inf. Circ. 6121, Bureau of Mines, 1929, 11 pp.

2 Keener, Oliver W., Methods and Costs of Mining at Hartley-Grantham Mine, Tri-State Zinc and Lead District: Inf. Circ. 6286, Bureau of Mines, 1930, 8 pp.

3 Poston, Roy H., Sinking Practices and Costs at the Pim Shaft, St. Louis Smelting and Refining Works of the National Lead Co., St. Francois, Mo.: Inf. Circ. 6588, 1932, 13 pp.

4 Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co. of New Mexico: Inf. Circ. 6368, Bureau of Mines, 1930, 21 pp.

5 Black, W. S., Cost of Shaft Sinking at Cripple Creek: Eng. and Min. Jour., vol. 120, Aug. 15, 1925, p. 255.

6 Jackson, Chas. F., and Knaebel, John B., Gold Mining and Milling in the United States and Canada, Current Practices and Costs: Bull. 363, Bureau of Mines, 1932, 151 pp.

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TABLE 3.- Typical shaft-sinking costs - Continued

Mine and location	Shaft section, feet	Kind of rock	Depth sunk, feet	Cost per foot	Remarks
Teck-Hughes, Ontario 7	6.33 by 21.33 12.5 by 13.0	Porphyry and lamprophyre	1,484	$96.34	-
		Porphyry	2,284	120.75	-
Magma, Ariz., No. 7 shaft 8	7.5 by 16.5	Schist	1,465	$75.24 total cost	Direct cost, $70.00.
Magma, Ariz., No. 5 shaft 9	7 by 21.5; 3 compartments	Do.	2,531	$116.44 total	Direct sinking cost, $76.75.
McPherson shaft, Tennessee 10	10 by 19	Schist and graywacke	200	65.34	1,600 to 1,800 levels.
Bisbee Queen, Ariz. 8	7 by 17	-	823	69.15	-

7 Henry, R. J., Mining Methods and Costs at the Teck-Hughes Gold Mines, Ltd., Kirkland Lake, Ontario: Inf. Circ. 6322, Bureau of Mines, 1930, 11 pp.

8 Gardner, E. D., and Johnson, J. Fred, Shaft-Sinking Practices and Costs: Bull. 357, Bureau of Mines, 1932, 104 pp.

9 Snow, Fred W., Mining Methods and Costs at the Magma Mine: Inf. Circ. 6168, Bureau of Mines, 1929, 32 pp.

10 Weaver, Lamar, Shaft Sinking in Tennessee: Eng. and Min. Jour., vol. 129, Mar. 24, 1930, p. 302.

TABLE 4.- Drifting and crosscutting data at lead and zinc mines

Mine and location	Type of ore and rock	Size of headings, feet	Mining method	Type of haulage	Cost per foot	Remarks
Mine No. 1, Tri-State district	Dolomite and chert.	7 by 7 (crosscuts between ore bodies termed "pull drifts").	Open stopes.	Hand, mules, and trolley locomotives.	$7.50 labor $14.13 total	Untimbered.
Mine No. 2, Tri-State district	Do.	Do.	Do.	Hand and mules	$6.50 labor only	Do.
Waco, Tri-State district	Do.	8 by 8	Do.	Do.	$6.00 to $8.00	Do.
Barr, Tri-State district	Do.	7 by 7 to 8 by 10	Do.	Do.	$6.00 labor $10.00 to $12.00 total	Do.
No. 8 mine, S.E. Missouri district	Dolomitic limestone.	7 by 8	Do.	Trolley locomotives.	$4.89 labor $7.87 total	Do.
Bonne Terre, S.E. Missouri district	Do.	7½ by 8 to 9 by 10 main haulageways 8 by 12 minimum.	Do.	Trolley locomotives, with mules for gathering.	-	Do.
No. 2 mine, Mascot, Tenn.	Do.	8 by 8 (main haulage) 7 by 6 (crosscuts).	Do.	Trolley locomotives.	$8.17 total	Ave. cost of drifts, crosscuts, raises, and slabbing.

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TABLE 4.- Drifting and crosscutting data at lead and zinc mines - Continued

Mine and location	Type of ore and rock	Size of headings, feet	Mining method	Type of haulage	Cost per foot	Remarks
Edwards, N.Y.	Dolomite and some gneiss.	6 by 7	Open stopes with some shrinkage.	Storage-battery locomotives.	-	Usually untimbered.
Bunker Hill, Idaho	Quartzite and ore.	7 by 8 (drifts) 15 by 8 (main crosscuts).	Squre sets.	Storage-battery locomotives on levels; trolley locomotives in tunnel.	$11.35	Ave. cost for 1,168 ft. in 1931..
Page mine, Idaho	Do.	6 by 8 inside timber, 8 by 9.5 rock dimension.	Do.	Storage-battery locomotives.	-	Drifts timbered.
Hecla mine, Idaho	Do.	8 by 8 clear for main haulage; 5 by 7 for crosscuts; 12 feet high in ore and full width of lode up to 18 feet.	Stull sets and fill.	Do.	-	-
Morning mine, Idaho	Do.	13 by 13 (drifts) 6 by 8 (crosscuts)	Do.	Do.	-	Drifts all timbered.
Silver King Coalition mines, Utah	Quartzite, limestone, and ore.	6 by 8 in clear.	Square sets.	Trolley locomotives on main levels.	$11.35 ave. for drifting, crosscutting, sinking, and raising.	Some timbered.
Park-Utah mine, Utah	Do.	8 by 8 (main haulage) 5 by 7 (exploration cross cuts.)	Square sets and cut-and-fill.	Trolley locomotives.	$23.84 main tunnel 8 by 8, 1/3 timbered.	Drifts usually timbered.
Tintic Standard mine, Utah	Limestone, quartzite, shale, and ore	6 by 8 over all 4.33 x 7.25 in clear.	Square sets.	Hand; mules for longer trams.	-	Timbered.
Black Rock mine, Montana	Granite and vein matter.	(6 by 8 to 8 by 10 over all) (3.66 by 7.25 to 5 by 7 in clear).	Do.	Storage-battery locomotives.	-	Drifts timbered.
Block P mine, Montana	Rhyolite and vein matter.	6 by 8 over all; 3.5 by 6.5 in clear.	Cut-and-fill.	Do.	-	Do.
Pecos mine, N. Mex.	Schist and ore.	7 by 8 (cross cuts) drifts 5.33 at top and 5.8 at bottom by 7 high in clear.	Cut-and-fill and square sets.	Do.	$8.08 labor $11.30 total.	Do.
Ground Hog mine, N. Mex.	Limestone, shale, sandstone, and ore	5 by 7	Square sets.	Hand.	$3.25 to $4.50 per foot for contract labor only.	Drifts not timbered.

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TABLE 5.- Tons of ore produced per foot of development work at lead and zinc mines

Mine or district	Type of ore body	Level interval, feet	Mining method	Tons produced per foot of development	Remarks
Tri-State district	Flat, irregular ore bodies.	Usually only one level.	Open stopes (heading and bench).	250	Average for 3 mines; all development included.
Do.	Do.	Do.	Do.	600	Average for 2 large mines; all development included.
No. 8 mine, S.E. Missouri district	Flat, irregular, bedded deposits.	One level.	Do.	250 plus	Estimated.
Mascot, Tenn.	Large, irregular, bedded deposits; dip, 18 to 22°.	50 to 75.	Open stopes, underhand mill-hole, and heading and bench.	155	-
Edwards, N.Y.	Irregular bedded deposits; dip, 40 to 45°.	Upper levels 100; lower levels 200.	Open stopes, with some shrinkage stoping.	27.5 (1930)	-
Bunker Hill mine, Idaho	Wide, deep, shear zone, large ore bodies.	200	Square sets and fill.	125	-
Page mine, Idaho	2 fissure veins in shear zone in quartzite; dip, 40 to 60°.	300	Do.	19	-
Morning mine, Idaho	Long fissure vein in quartzite; continuous to great depth; dip, 80° to vertical.	200	Stringer sets and fill.	66	-
Silver King Coalition mines, Utah	Large, irregular bedded deposits in limestone; some fissures in quartzite.	200 at lower levels.	Square sets and fill.	5.4	-
Park-Utah, Utah	Fissures in quartzite and limestone; ore shoots, 3 to 80 feet wide up to 900 feet long; dip, 40 to 55°.	200	Square sets and cut-and-fill.	15.7	-
Pecos Mine, N. Mex. 1	Irregular lenses in shear zone in schist.	100	Cut-and-fill and square sets; some shrinkage.	14.7	-
Ground Hog, N. Mex.	Ore body 3 to 25 feet wide by 600 feet long in fault fissure; dip, 50°.	100	Square sets and fill.	13.2	-

1 1928-30.

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TABLE 6.- Current mine development costs per ton of ore mined

Mine and period covered	Type of orebody	Level interval, feet	Mining method	Tons ore mined during period	Cost of development per ton of ore mined
No. 8 mine, S.E. Missouri district 1	Irregular, flat-bedded deposits.	One level	Open stopes (heading and bench).	168,089	$0.045
Mascot, Tenn. 2	Large, irregular-bedded deposits; dip, 18-22°.	50 to 75.	Open stopes, underhand mill-hole, heading and bench.	528,626	.053
Page mine, Idaho 1	2 fissure veins in shear zone in quartzite; dip, 40 to 60°.	300	Do.	90,460	1.170
Morning mine, Idaho 1	Large fissure vein in quartzite; continuous to great depth.	200	Stringer sets and fill.	276,890	.442
Hecla and Star mines, Idaho 1	Several ore bodies in shear zone; dip, 70 to 80°.	300 and 400	Do.	312,942	.638
Silver King Coalition, Utah 2	Large, irregular-bedded deposits in limestone; some veins in quartzite.	200 at lower levels.	Square set and fill.	180,208	2.086 (development in rock and ore).
Park-Utah 1	Fissures in quartzite and limestone; ore shoots, 3 to 80 feet wide, up to 900 feet long; dip, 40 to 55°.	200	Square sets and cut-and-fill.	192,250	.828 (development in rock and ore).
Tintic Standard, Utah 3	Trough ore bodies replacing limestone beds; extend over wide areas and are of large vertical height.	Usually 100 (200 feet 900 to 1,100; 150 feet 1,100 to 1,250).	Square set and fill.	8,758	.299 in ore .844 in rock 1.143 total.
Pecos mine, N. Mex. 4	Irregular lenses in shear zone in schist.	100	Square set and fill and cut-and-fill; some shrinkage.	578,658	All development, including shaft sinking, stations and pockets $0.859.
Ground Hog, N. Mex. 5	Ore body, 3 to 25 feet wide by 600 feet long in fault fissure; dip, 50°.	100	Square set and fill.	11,146	.922 in ore .471 in rock 1.393 total.

1 1928.

2 1929.

3 November 1929.

4 1927-29.

5 3 months, 1930.

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STOPING

At most mines development work continues long after stoping is begun, and in some mines stoping starts very early in the development period. Thus, in the Tri-State district, where stoping often is begun almost immediately after the shaft is bottomed, the ore is developed as it is stoped although it has been previously delimited, at least in part, by churn drilling.

In other districts, where the deposits are of a different type, considerable development is required to prepare the ore bodies for stoping. Where enough capital is available, good judgment usually dictates that during the early life of a mine development be carried on at a rate which will block out ore faster than it is removed by stoping, thus providing a substantial reserve of ore. Such a reserve not only serves as a backlog, assuring continuous and even production, but forms a basis for planning for the most efficient operation. On the other hand, development too far ahead of ore extraction serves only to absorb excessive capital and interest charges.

Selection of Stoping Method

The stoping method employed at most mines has been evolved from experiences during their earlier life, at least as to the variations in the general method. Low stoping costs obviously are desirable, although the method resulting in the lowest cost per ton of ore will not always give the lowest cost per pound of lead and zinc. Therefore a higher cost-per-ton method may be preferable to a lower cost-per-ton method in some instances. Some stoping methods afford much lower costs than others, but physical conditions of the deposit limit the operator in his selection. In some instances the proper method to employ is immediately apparent, whereas in others a real problem is involved in determining that which is most suitable.

During 1930, lead and zinc mines in the United States produced roughly 19,400,000 tons of crude ore, of which about 14,661,000 tons were mined from open stopes, 1,896,000 tons by square-set stoping, 1,359,000 tons by cut-and-fill stoping (including 710,000 tons by stringer-set and fill in Idaho), 974,000 tons by shrinkage stoping, 476,000 tons by top slicing, and 34,000 tons by block caving.

In the bulletin manuscript, each of these methods is discussed in detail, and the stoping operations at a large number of different mines are described and illustrated. In addition, the advantages and disadvantages of each stoping method and the conditions to which each is adapted were discussed.

Cost of Stoping

The cost of stoping varies widely with the physical characteristics of the deposit; its size, shape, dip, strength of ore and wall rocks; method of stoping, grade of ore, and regularity of its occurrence, rate of production, and skill of the management. Where ore and waste must be sorted in the stopes the cost per ton of ore mined usually is increased considerably, though often this results in lowering the cost per pound of lead and zinc, which is the end to be sought. Since conditions are rarely, if ever, identical at two different mines, obviously direct comparisons of cost between any two mines are apt to be misleading. In table 7, stoping costs (in dollars) at a number of lead and zinc mines are summarized; and stoping costs in units of labor, explosives, timber, and power are given in table 8. In these tables, direct stoping costs only are given, and costs of development and underground transportation, as well as general underground and surface expense, have not been included.

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COST OF MINING

In this section, total direct mining costs are discussed. These costs, which include development, stoping, underground transportation, and general charges, are given for a number of mines in tables 9, 10, and 11. Differences in accounting practices at the different mines make it difficult to distribute the costs between the several items in the tables on exactly the same basis, but the figures have been compiled on as nearly a uniform basis as possible. In table 12, total direct mining costs are presented in terms of labor, materials, and power.

TABLE 7.- Stoping costs, lead and zinc mines

Mine and location	Type of deposit	Period covered	Ore hoisted during period, tons	Stoping method	Direct stoping cost per ton ore hoisted	Remarks
Mine No. 1, Tri-State district	Flat-bedded deposit.	6 months, 1928.	68,770	Open stopes.	$0.532	$0.049 deducted from reported mining cost for development.
Mine No. 2, Tri-State district	Do.	9 months, 1928.	87,563	Do.	.458	$0.050 deducted for development.
Waco mine, Tri-State district	Do.	May 1927, to Jan. 1929.	364,285	Do.	.430	Do.
Barr mine, Tri-State district	Do.	Jan. to April 1929	45,344	Do.	.497	$0.047 deducted for development.
Hartley-Grantham mine, Tri-State district	Do.	Jan. to June 1929	60,282	Do.	.516	-
Hartley mine, Tri-State district	Do.	6 months, 1930.	180,212	Do.	.572	-
No. 8 mine, S.E. Missouri district	Flat-bedded deposits, disseminated-lead ore.	Year 1928	168,089	Do.	.483	-
Mascot No. 2 mine, Mascot, Tenn.	Large, bedded deposits; dip, 18 - 22°.	10 months, 1929.	511,383	Open stopes; mill-hole system.	.222	Based on ore from stopes only.
Bunker Hill & Sullivan mine, Idaho	Wide, deep, shear zone, large ore bodies.	Year 1928	452,345	Square-set and fill.	2.231	$0.13 deducted from reported cost for transportation.
Do.	Do.	Year 1931	460,366	Do.	2.476	$0.134 deducted for transportation and $0.518 for distributed charges.
Page mine, Idaho	2 fissure veins in shear zone; dip, 40 to 60°.	Year 1928	90,460	Square-set and fill.	2.15	Do.
Silver-King Coalition mines, Utah	Large, irregular bedded deposits in limestone; some veins in quartzite.	Year 1929	180,208	Do.	3.50	-

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TABLE 7.- Stoping costs, lead and zinc mines - Continued

Mine and location	Type of deposit	Period covered	Ore hoisted during period, tons	Stoping method	Direct stoping cost per ton ore hoisted	Remarks
Park-Utah mine, Utah	Fissures in quartzite and limestone.	Year 1928	192,250	Square-set and fill; some cut-and-fill.	$2.426	-
Tintic Standard mine, Utah	Trough ore bodies replacing limestone beds.	November 1929.	8,758	Square-set and fill.	4.973	3 types of ore must be mined separately, and considerable sorting is required.
Ground Hog mine, New Mexico	Ore body 3 to 25 feet wide by 600 feet long in fault fissure.	3 months, 1930.	11,146	Square-set and fill and timbered rill.	1.762	-
Pecos mine, New Mexico	Irregular lenses in shear zone in schist.	1927-28-29	194,030	Square-sets.	2.66	Based on ore won by square-setting only.
Do.	Do.	Do.	270,873	Cut-and-fill.	2.11	Based on ore won by cut-and-fill stoping only.
Do.	Do.	Do.	23,806	Shrinkage and subsequent fill.	.93	Based on ore won by shrinkage stoping only.
Morning mine, Idaho	Large fissure vein in quartzite, continuous to great depth.	Year 1928	276,890	Stringer set and fill.	2.371	-
Hecla and Star mines, Idaho	Several ore bodies in shear zone; dip, 70 to 80°.	Year 1928	312,942	Stringer set and fill principally.	1.211	-

TABLE 8.- Stoping costs in units of labor, explosives, timber, and power

Mine and location	Type of deposit	Period covered	Tons ore hoisted during period	Stoping method	Unit costs per ton ore, stoping only			
					Man-hours	Explosives, pounds	Timber	Power, kw.h.
Mine No. 1, Tri-State district	Flat-bedded deposits	6 months, 1928.	68,770	Open stopes.	0.784	0.750	None	6.00
Mine No. 2, Tri-State district	Do.	9 months, 1928.	87,563	Do.	.642	1.265	Do.	2.51
Mine No. 3, Tri-State district	Do.	Year 1927	132,384	Do.	.401	.875	Do.	1.40
Waco mine, Tri-State district	Do.	May 1927 to Jan. 1929	364,285	Do.	.635	.594	None	-
Barr mine, Tri-State district	Do.	January to April 1929	45,344	Do.	.502	.805	Do.	2.16

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TABLE 8.- Stoping costs in units of labor, explosives, timber, and power - Continued

Mine and location	Type of deposit	Period covered	Tons ore hoisted during period	Stoping method	Unit costs per ton ore, stoping only			
					Man-hours	Explosives, pounds	Timber	Power, kw.h.
Hartley-Grantham mine, Tri-State district	Flat-bedded deposits.	January to June 1929	60,282	Open stopes.	.490	.741	None	-
Hartley mine, Tri-State district	Do.	6 months, 1930.	180,212	Do.	.460	1.120	Do.	2.00
No. 8 mine, S.E. Missouri district	Flat-bedded deposits, disseminated-lead ore.	Year 1928	168,089	Do.	.441	.565	Do.	-
Mine X, S.E. Missouri district	Do.	Ave. month, 1929.	49,700 Stope ore only	Do.	.673	.459	Do.	4.1
Mascot No. 2 mine, Mascot, Tenn.	Large bedded deposits; dip, 18 to 22°.	10 months, 1929.	511,383 Stope ore only	Open stopes, mill-hole system.	.316	.502	Do.	3.95
Edwards, N.Y.	Lenticular masses in folded beds.	Year 1930	72,742 Stope ore only	Open stopes.	.928	?	?	21.16
Silver King Coalition mine,	Large, irregular-bedded deposits in limestone.	Year 1929	180,208	Square set and fill.	3.920	?	?	?
Park Utah mine, Utah	Fissures in quartzite and limestone.	Year 1928	192,250	Square set and fill; some cut-and-fill.	1.930	?	?	?
Tintic Standard mine, Utah	Trough ore bodies, replacing limestone beds.	November 1929.	8,758	Square set and fill.	4.694	.500	24.03 bd.ft.	26.47
Pecos mine, New Mexico	Irregular lenses in shear zone in schist.	Year 1929	195,685 stoping only 217,121 total ore	Square set and fill, cut-and-fill, pillars and shrinkage.	1 3.479 2 3.135	1 .730 2 .658	1 3.9 linear feet. 2 3.5 linear feet.	1 6.97 2 6.28
Morning mine, Idaho	Large fissure vein in quartzite.	Year 1928	276,890	Stringer sets and fill.	-	.693	-	-
Hecla and Star mines, Idaho	Several ore bodies in shear zone; dip, 70 to 80°.	Year 1928.	312,942 total tonnage; all	Stringer sets and fill; horizontal stoping.	1.318	.533	1.234 linear feet stulls	-

1 Based on tonnage of ore from stopes only.

2 Based on total ore hoisted.

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TABLE 8.- Stoping costs in units of labor, explosives, timber, and power - Concluded

Mine and location	Type of deposit	Period covered	Tons ore hoisted during period	Stoping method	Unit costs per ton ore, storing only			
					Man-hours	Explosives, pounds	Timber	Power, kw.h.
Hecla and Star mines, Idaho	Several ore bodies in shear zone; dip, 70 to 80°.	Year 1928.	-	Timbered shrinkage and horizontal stringer sets and fill.	1.394	1.174	11.207 bd. ft. sawed timber.	12.66
Do.		Do.	-	Shrinkage, West ore body.	1.410	2.054		-
Block P mine, Montana	Narrow fissure; dip, 65 to 80°.	Year 1929.	106,242	Cut-and-fill.	3.956	2.568	-	-
Gilman, Colo.	Replacement ore bodies in lime beds.	-	-	Ave. of 11 square-set stopes.	1.724	.67	13.9 bd.ft.	-
Do.	Do.	-	-	Ave. of 9 cut-and-fill stopes.	1.250	.53	7.3 bd.ft.	-
Do.	Do.	-	-	Open stope.	.879	.38	3.5 bd.ft.	-
Do.	Do.	-	-	Underhand pillar stopes.	1.250	.41	6.7 bd.ft.	-

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TABLE 9.- Mining costs per ton of ore hoisted

Mine and location	Period covered	Tons ore produced during period	Stoping method	Costs of mining per ton of ore					
				Development and exploration	Stoping	Transportation	General underground expense	General surface expense	Total
Mine No. 1, Tri-State district	6 months, 1928.	68,770	Open stopes.	$0.049	$0.532	$0.217	($0.187)		$0.985
Mine No. 2, Tri-State district	9 months, 1928.	87,563	Do.	.050	.458	.200	$.165	$0.120	.993
Mine No. 3, Tri-State district	Year 1927.	132,384	Do.	.030	.450	.165	.162	.194	1.001
Waco mine, Tri-State district	May 1927 to January 1929.	364,285	Do.	.050	.490	.155	.057	.033	.785
Barr mine, Tri-State district	January to April 1929.	45,344	Do.	.047	.497	.217	.069	.032	.862
Hartley-Grantham mine, Tri-State district	January to June 1929.	60,282	Do.	None.	.516	.197	.049	.020	.782
Hartley mine, Tri-State district	6 months, 1930.	180,212	Do.	Do.	.572	.152	.063	.041	.828
No. 8 mine, S.E. Missouri district	Year 1928.	168,089	Do.	.045	.483	.222	(.136)		.886
Mascot No. 2 mine, Tennessee	10 months, 1929.	528,626	Open stopes; mill-hole system.	.053	.222	.278	(.086)		.639
Bunker Hill & Sullivan mine, Idaho	Year 1928.	452,345	Square-set and fill.	[1].053	[2]2.231	[3].381	1.037	.195	3.897
Do.	Year 1931.	460,366	Do.	[4].066	[5]2.476	[6].308	([7].604)		3.454
Page mine, Idaho	Year 1928.	90,460	Do.	1.170	2.150	.160	.180	.160	3.820
Silver King Coalition mines, Utah	Year 1929.	180,208	Do.	2.086	3.502	1.848	.309		7.745

[1]Difference between mining reported as $2.414 and stoping reported as $2.361. Probably represents exploration only.
[2]Reported, $2.361; deducted $0.130 for carmen, trammers and hoistmen. Probably includes some development.
[3]Reported, $0.251; $0.130 added for carmen, trammers and hoistmen reported under stoping.
[4]Reported as exploration.
[5]Reported, $3.128; deducted $0.134 for carmen, trammers, and hoistmen and $0.518 distributed charges. Probably includes some devel't.
[6]Reported $0.174; $0.134 added for carmen, trammers, and hoistmen.
[7]$0.518 distributed charges reported under stoping, plus pumping cost.

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TABLE 9.- Mining costs per ton of ore hoisted - Continued

Mine and location	Period covered	Tons ore produced during period	Stoping method	Costs of mining per ton of ore					
				Development and exploration	Stoping	Transportation	General underground expense	General surface expense	Total
Park Utah mine, Utah	Year 1928.	192,250	Square-set and fill; some cut-and-fill.	$0.828	$2.426	$0.282	$0.583	$0.543	$4.662
Tintic Standard mine, Utah	November 1929.	8,758	Square-set and fill.	1.143	4.973	1.158	.322	[8]1.888	9.484
Ground Hog mine, New Mexico	3 months, 1930.	11,146	Square-set and fill and timbered rill.	1.393	1.762	.628	.080	.127	3.990
Pecos mine, New Mexico	Years 1927-29.	578,658	Square-set and cut-and-fill.	[9]1.005	2.278	.663	.423	.010	4.379
Morning mine, Idaho	Year 1928.	276,890	Stringer sets and fill.	.442	2.371	.530	.698	.204	4.245
Hecla and Star mines, Idaho	Year 1928.	312,942	Do.	.638	1.211	.322	.766	.360	3.297

[8]Includes a number of charges probably not included in costs given for most of the other mines, such as unloading, yard maintenance, loading, and shipping $0.218; building maintenance, $0.248; heat, water, and light, $0.261.
[9]Includes $0.146 for diamond drilling.

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TABLE 10.- Mining costs per ton of ore hoisted

Mine and location	Period covered	Ore produced during period, tons	Stoping method	Direct costs of mining per ton of ore						
				Labor and supervision	Compressed air, drills, and steel	Power	Explosives	Timber	Other supplies	Total
Mine No. 1, Tri-State district	6 months, 1928.	68,770	Open stopes.	$0.624	$0.127	$0.041	$0.110	.....	$0.083	$0.985
Mine No. 2, Tri-State district	9 months, 1928.	87,563	Do.	.576	.114	.076	.157	.....	.070	.993
Mine No. 3, Tri-State district	Year 1927.	132,384	Do.	.675	.075	.102	.117	.....	.032	1.001
Waco mine, Tri-State district	May 1927 to January 1929.	364,285	Do.	.536	.068	.109	.072	.....	.....	.785
Barr mine, Tri-State district	January to April 1929.	45,344	Do.	.529	.066	.048	.103	.....	.116	.862
Hartley-Grantham mine, Tri-State district	January to June 1929.	60,282	Do.	.535	.052	.037	.106	.....	.051	.781
Hartley mine, Tri-State district	6 months, 1930.	180,212	Do.	.507	.078	.060	.150	.....	.033	.828
No. 8 mine, S.E. Missouri district	Year 1928.	168,089	Do.	.687	.081	.046	.072	.....	.....	.886
Mascot No. 2 mine, Mascot, Tenn.	10 months, 1929.	528,626	Open stopes, mill-hole system.	.454	.047	.016	.060	.....	.062	.639
Bunker Hill & Sullivan mine, Idaho	Year 1928.	452,345	Ssquare-set and fill.	1.934	.229	(1)	.167	$0.321	1.246	3.897
Do.	Year 1931.	460,366	Do.	1.875	.204	.113	.165	.331	.766	3.454
Page mine, Idaho	Year 1928.	90,460	Do.	2.690	.280	.120	.240	.380	.110	3.820
Silver King Coalition mines, Utah	Year 1929.	180,208	Do.	5.418	.854	.370	.486	.340	.277	7.745
Park Utah mine, Utah	Year 1928.	192,250	Square-set and fill; some cut-and-fill.	3.062	.139	.191	.227	.493	.550	4.662

[1]Included in other items.

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TABLE 10.- Mining costs per ton of ore hoisted - Continued

Mine and location	Period covered	Ore produced during period, tons	Stoping method	Direct costs of mining per ton of ore						
				Labor and supervision	Compressed air, drills, and steel	Power	Explosives	Timber	Other supplies	Total
Tintic Standard mine, Utah	November 1929.	8,758	Square-set and fill.	$6.356	$0.770	$0.261	$0.224	$0.913	$0.960	[2]9.484
Ground Hog mine, New Mexico	3 months, 1930.	11,146	Do.	2.793	.169	.179	.187	.318	.344	3.990
Pecos mine, New Mexico	Year 1929.	217,121	Square-set and fill and cut-and-fill.	3.308	.141	.091	.259	.422	.677	4.898
Morning mine, Idaho	Year 1928.	276,890	Stringer sets and fill.	3.088	.122	.211	.158	.417	.249	4.245
Hecla and Star mines, Idaho	Do.	312,942	Do.	1.868	.070	.145	.112	.357	.317	2.869

[2]Includes some surface charges, probably not included in costs given for most of the other mines listed, such as unloading, yard maintenance, loading, and shipping, $0.218; building maintenance, $0.248; heat, water, and light, $0.261.

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TABLE 11.- Tons of ore per man-shift, 1930, lead and zinc mines

Number of mines reporting	Ore produced, tons	Stoping method employed	Man-shifts worked			Ore produced per man-shift, tons		
			Underground	Surface	Total	Underground	Surface	Total
102	13,287,051	Open stopes.	1,676,562	226,780	1,903,342	7.925	58.6	6.98
26	1,716,695	Square-sets.	1,106,917	208,886	1,315,803	1.551	8.2	1.30
13	1,098,823	Cut-and-fill and stringer-set and fill.	450,079	120,965	571,044	2.441	9.1	1.92
7	651,155	Shrinkage.	184,554	57,155	241,709	3.528	11.4	2.69
Total 148	16,753,724		3,418,112	613,786	4,031,898	4.902	22.3	4.15

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Mine and location	Period covered	Ore produced during period, tons	Stoping method	Unit costs per ton of ore produced					
				Labor, man-hours			Explosives, pounds	Power, kw. h.	Timber
				Development	Stoping only[1]	Total[2]			
Mine No. 1, Tri-State district	6 months, 1928.	68,770	Open stopes.	.....	0.784	1.341	0.750	7.43	None.
Mine No. 2, Tri-State district	9 months, 1928.	87,563	Do.	.....	.642	.876	1.265	5.75	Do.
Mine No. 3, Tri-State district	Year 1927.	132,384	Do.	.....	.401	.905	.875	4.17	Do.
Waco mine, Tri-State district	May 1927 to January 1929.	364,285	Do.	.....	.635	1.041	.594	4.99	Do.
Barr mine, Tri-State district	January to April 1929.	45,344	Do.	.....	.502	.888	.805	6.09	Do.
Hartley-Grantham mine, Tri-State district	January to June 1929.	60,282	Do.	.....	.490	.745	.741	.....	Do.
Hartley mine, Tri-State district	6 months, 1930.	180,212	Do.	.....	.460	.673	1.120	4.25	Do.
No. 8 mine, S.E. Missouri district	Year 1928.	168,089	Do.	.....	.490	.875	.634	4.56	Do.
Mine X, S.E. Missouri district	Ave.month, 1929.	[3]56,000	Do.	0.135	.650	.827	.550	10.30	Do.
Mascot No. 2 mine, Tenn.	10 months, 1929	528,626	Do.	.....	.316	.831	.502	9.68	Do.
Edwards, New York	Year 1930.	[4]72,742	Do.	.053	.928	1.503	.....	61.09	?
Silver King Coalition mines, Utah	Year 1929.	180,208	Square-set and fill.	.....	.....	8.055	3.607	77.70	?
Park Utah mine, Utah	Year 1928.	192,250	Square-set and fill; some cut-and-fill.	0.580	1.930	3.540	1.530	29.23	16.46 bd.ft.
Tintic Standard mine, Utah	November 1929.	8,758	Square-set and fill.	1.038	4.694	7.225	1.068	46.60	25.50 bd.ft.
Pecos mine, New Mexico	Year 1929.	[5]217,121	Square-set and fill; cut-and-fill; pillars and shrinkage.	[6]1.463	[6]3.135	[6]5.674	[6]1.338	[6]13.93	4.7 linear fet.
Morning mine, Idaho	Year 1928.	276,890	Stringer sets and fill.	.....	.....	4.169	.964	39.37	12.75 bd.ft.
Hecla and Star mines, Idaho	Do.	312,942	Do.	.....	.....	2.029	.783	17.01	1.234 linear feet plus 11.21 bd.ft.
Block P mine, Montana	Year 1929.	106,242	Cut-and-fill.	.284	3.956	[7]5.350 [8].469 5.819	2.854	17.46	7.538 linear feet plus 3.955bd.ft.

[1]Does not include tramming, haulage, hoisting and general. [2]Includes tramming, hauage, hoisting and general. [3]Includes 2,000 tns of waste rock. [4]Not including ore from development. [5]Includes stope ore 195,685 tons and development ore 21,436 tons. [6]Based on total tonnage. [7]Underground. [8]Surface.

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PART 3

METHODS OF CONCENTRATING LEAD AND ZINC ORES

This section is a summary of a number of information circulars dealing with concentration of ores at various lead and zinc mines in the United States.

Many changes have been made during the past 15 or 20 years in methods of concentation, with a resulting marked improvement in recoveries and costs. The earlier types of grinding equipment have been replaced largely by rod or ball mills; flotation has superseded gravity-type concentrators or has been added to the flow sheet in many plants; differential flotation has permitted the separation of lead from zinc minerals in finely disseminated form; and screening and classification have been improved.

The bulletin manuscript includes summaries covering concentration methods at each of 17 plants, giving details regarding the nature of the ore treated; plant capacity; milling methods; grade of ore, concentrates, and tailings; recoveries; ratio of concentration; and water consumption. Flow sheets of each plant are given.

Key to concentrators listed in tables and referred to in text

	Information Circular
A. Netta Mill, Picher, Okla.	6342
B. White Bird Concentrator, Picher, Okla.	6353
C. Bunker Hill & Sullivan West Mill, Idaho	6314
D. Hecla Concentrator, Gem, Idaho	6600
E. Mascot Concentrator, Mascot, Tenn.	6379
F. Flat River, Missouri Mill	6658
G. Page Concentrator, Kellogg, Idaho	6590
H. Morning Concentrator, Mullan, Idaho	6587
I. Hughesville, Mont., Concentrator	6447
J. Tybo, Nev., Concentrator	6430
K. Midvale, Utah, Concentrator	6492
L. Chief Consolidated Concentrator, Eureka, Utah	6320
M. Montana Mine Concentrator, Ruby, Ariz.	6497
N. Black Hawk Concentrator, Hanover, N. Mex.	6359
O. Pecos Concentrator, Tererro, N. Mex.	6605
P. Balmat Mill, St. Lawrence County, N.Y.	6574

General

Thorough study of the composition, mineral distribution, physical characteristics, amenability to gravity and flotation treatment, and other determining factors is necessary to prescribe the best method of treatment for a given lead-zinc ore. Local conditions, as well as the character and grade of products desired with reference to smelting contracts, often have an important bearing upon the method of treatment. Since fairly universal acceptance of the flotation process as a means for better separations and improved recoveries in the treatment of slimes and finely ground ore, many complex and finely disseminated sulphide ores can now be treated successfully which will not yield acceptable products by gravity methods. It might be said that the Tri-State, Southeastern Missouri, and Mascot ores examplify those which can be treated suitably by gravity or combined gravity and flotation methods, whereas the ores of the Coeur d'Alene and some other western districts are, in most cases, suited to all-flotation methods.

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Hand Sorting

Hand sorting is practiced at relatively few lead-zinc plants, although both waste rock and crude lead-smelter product are sorted by hand at concentrator D. At other places considerable waste is sorted from the ore in the mines.

Crushing

As a rule, the ore delivered to the crushing plant is broken to pass a grizzly with 10-inch or smaller openings. The ore is first crushed in jaw crushers or gyratories to about 2 or 3 inches, although at concentrator E the crusher discharges a product of 5½ inches maximum size and at concentrator D the jaw crusher is set at 5 inches. In the Tri-State district the crusher product usually is fed directly to rolls, but in other districts it is first passed over stationary, trommel, or vibrating screens with ½- to 2-inch openings; the oversize is fed to disk or cone crushers, gyratories, or rolls, in closed circuit with the screen. In plants where gravity concentration is employed the jig and table middlings generally are reground in secondary rolls, or in ball or rod mills, for treatment by sand jigs, tables, or flotation. Typical screen analyses of products from crushing plants are given in tables 13, 14, and 15.

Grinding

Rod and ball mills have largely replaced other types of mills for liberating mineral particles in middlings and finely disseminated ores. The feed to these mills may comprise middlings from sand jigs and tables or oversize from fine screens, and tailings from tables for further treatment on tables; all middlings and tailings from jigs and tables, or tailings from sand and cleaner jigs only, for further treatment by flotation; or, where no gravity concentration is employed, it may consist of the mill feed that has been crushed and, after grinding, is subjected to all-flotation treatment. In nearly all instances the discharge is classified by drag classifiers, the sands either are returned to the grinding mill or to hydraulic classifiers, and the overflow goes to thickeners for subsequent flotation treatment. The screen analyses in tables 16 to 19 indicate the fineness of grinding at several plants.

TABLE 13.- Screen analyses of successive crusher products: concentrator F

Size	Weight, percent			
	Gyratory-crusher discharge (Symons-crusher feed)	Symons horizontal-disk-crusher discharge (new feed to rolls	Composite roll feed (new feed and dry screen oversize)	Roll discharge
+3.0-inch.....	11.7	-	-	-
2.1-inch.....	19.0	-	-	-
1.05-inch....	29.6	45.5	6.2	-
.0525-inch	15.0	34.3	11.9	8.9
3-mesh.....	9.5	11.3	40.2	35.6
8-mesh.....	7.9	4.5	37.1	39.9
20-mesh.....	3.5	1.6	2.7	5.8
48-mesh.....	1.6	.8	.4	2.6
200-mesh.....	1.2	.7	.4	1.7
-200-mesh.....	1.0	1.3	1.2	5.5
Total.....	100.0	100.0	100.0	100.0

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TABLE 14.- Screen analyses of crusher products: concentrator G

Size	Weight, percent				
	Mine-run ore (feed to Blake crusher)	Blake-crusher discharge	Feed to Symons-cone crusher	Symons-cone-crusher discharge (feed to rolls)	Roll discharge
+3.0-inch..	18.0	2.67	1.60	-	-
2.0-inch..	6.33	6.25	14.50	-	-
1.0-inch..	16.67	11.11	45.60	18.82	-
.5-inch..	12.33	10.67	27.40	45.88	4.7
4-mesh.....	17.97	17.50	2.50	19.80	25.4
8-mesh.....	7.30	11.20	.60	4.10	17.1
20-mesh.....	6.90	16.00	1.40	2.80	17.1
4-mesh.....	4.80	8.10	1.80	2.30	10.7
200-mesh.....	3.50	6.80	1.50	2.10	7.6
-200-mesh.....	6.20	9.70	3.10	4.30	17.4
Totals.....	100	100	100	100.10	100

TABLE 15.- Screen analysis of crusher and roll products: concentrator L

Size	Weight, percent	
	Gyratory discharge (feed to rolls)	Roll discharge
+2.0-inch....	16.006	-
1.05-inch..	25.529	7.407
.525-inch	17.572	24.779
3-mesh.....	19.758	22.842
8-mesh.....	5.944	19.339
20-mesh.....	5.561	9.547
48-mesh.....	4.397	7.012
200-mesh.....	4.006	6.813
-200-mesh.....	1.227	2.261

Table 20 shows some of the conditions under which rod and ball mills are operated. It will be noted that at plants where all-flotation is employed the grinding mills are operated with a pulp density of 70 to 85 ercent solids.

Steel Consumption

The consumption of steel balls and rods averages 2 to 3 pounds per ton of ore ground and that of the liners 0.4 to 0.5 pound.

Sizing and Classifying

Grading of crushed material into different sizes usually is accomplished by trommel screens for the coarser jig sizes and by vibrating screens for the finer jig and table sizes.

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TABLE 16.- Screen analyses of ball-mill products grinding low-grade middlings in closed circuit with drag classifier; concentrator C

Size, mesh	Weight, percent			
	Feed to ball mill	Ball-mill discharge	Drag classifier Sand	Drag classifier Overflow
+3	0.9	-	-	-
8	51.4	2.2	1.0	-
20	27.7	6.0	5.4	-
48	11.1	24.8	31.4	-
100	7.7	32.2	39.0	6.5
200	1.1	14.9	14.3	21.4
-200	.1	19.9	8.9	72.1
Total	100.0	100.0	100.0	100.0

TABLE 17.- Screen analyses of rod-mill products grinding screen oversize and low-grade table middlings in open circuit; concentrator F

Size, mesh	Weight, percent			
	Screen oversize (1)	Low-grade middlings from drag classifier (2)	Screen oversize and low-grade table middlings	
			Rod-mill feed (1) and (2)	Rod-mill discharge
+8	31.6	0.3	8.3	-
20	53.9	23.8	39.0	0.5
48	5.4	58.5	43.0	43.9
100	1.9	15.3	4.5	19.7
200	1.4	.8	1.2	8.7
-200	5.8	.5	4.0	27.2
Total	100.0	99.2	100.0	100.0

TABLE 18.- Screen analyses of rod mill products grinding mill feed crushed to 1-inch size in closed circuit with drag classifier, concentrator I

Size	Weight, percent			
	Rod-mill feed	Classifier feed	Classifier sand	Classifier overflow
+1-inch	C.7	-	-	-
0.5-inch	12.1	-	-	-
3-mesh	37.2	-	-	-
6-mesh	16.9	-	-	-
20-mesh	19.6	0.4	C.6	-
48-mesh	8.1	14.5	18.3	1.1
100-mesh	2.8	37.7	41.8	23.7
200-mesh)	2.6	28.6	30.5	27.7
-200-mesh)		18.8	8.8	47.5
Total	100.0	100.0	100.0	100.0

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TABLE 19.- Screen analyses of ball-mill products in closed circuit with drag classifier; concentrator L

Size	Weight, percent			
	Ball-mill feed	Ball-mill discharge	Classifier sands	Classifier overflow
+1.05-inch	7.41	-	-	-
0.525-inch	24.78	-	-	-
3-mesh	22.84	-	-	-
8-mesh	19.34	-	-	-
20-mesh	9.55	6.8	7.0	-
48-mesh	7.01	32.3	38.3	-
100-mesh	3.90	25.4	34.4	10.89
200-mesh	2.91	10.3	10.1	15.84
-200-mesh	2 26	25.2	10.2	73.27
Total	100.00	100.0	100.0	100.00

TABLE 20.- Operating data of ball and rod mills at various lead and zinc concentrators

Concentrator	Number and size of grinding mill	Feed to grinding unit	Maximum and minimum size in feed, percent	Maximum and minimum size in discharge, percent	Pulp density of discharge, percent solid
A	1, 6-ft. ball mill.	De-watered bed middlings from De Mier level jig and zinc-sand middlings from tables.	31.6 on 20-mesh, 3.65 minus 65-mesh.	4.44 on 20-mesh, 40.67 minus 65-mesh.	-
B	1, 6-ft. by 36-inch ball mill.	Oversize from 2mm. screen ahead of tables and dewatered tailings from coarse-sand tables.	Max. 1/8-in.	1.7 on 14-mesh, 17.9 minus 100-mesh.	70
C	1, 8-ft. by 36-in. and 3, 6-ft. by 22-in. ball mills.	Dewatered middlings and tailings from jigs and tables. Low-grade circuit.	13.1 on 4-mesh, 0.1 minus 200-mesh.	0.5 on 4-mesh, 19.9 minus 200-mesh.	70.2
H	7, 8-ft. by 36-in. and 1, 8-ft. by 22-in. ball mills.	Mill feet to 5 primary mills in partial closed circuit with classifier, part of sands returning to primary mills and part to 2 secondary mills. One 8-ft. by 22-in. mill for re-grinding middlings.	Small percent on 1 in.	-	-

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TABLE 20.- Operating data of ball and rod mills at various lead and zinc concentrators - Continued

Concentrator	Number and size of grinding mill	Feed to grinding unit	Maximum and minimum size in feed, percent	Maximum and minimum size in discharge, percent	Pulp density of discharge, percent solids
I	1, 6-ft. 6-in. by 12-ft. 1-in. rod mill.	Mill feed. Rod mill in closed circuit with classifier.	12-8 on 1/2-in., 2.6 minus 100-mesh.	0.4 on 20-mesh, 18.8 minus 200-mesh.	-
J	2, 8-ft. by 48-in. ball mills.	Mill feed to ball mills in closed circuit with classifiers.	Minus 1/2-in.	9.75 on 14-mesh, 37.77 minus 200-mesh.	75 to 78
K	3, 5-ft. by 10-ft. rod mills and 3 5-ft. by 10-ft. ball mills.	Mill feed to 3 rod mills in closed circuit with classifier. Overflow to second classifier in closed circuit with 3 ball mills.	Rod mills: 6.1 on 1-in., 12.6 minus 200-mesh. Ball mills: 0.2 on 14-mesh, 11.2 minus 200-mesh.	Rod mills: 0.4 on 14-mesh, 27.2 minus 200-mesh. Ball mills: 0.2 on 14-mesh, 25.7 minus 200-mesh.	75 to 80
L	1, 8-ft. by 48-in. and 1, 8-ft. by 36-in. ball mill.	Mill feed to ball mills in closed circuit with classifier.	7.407 on 1.05-in., 2.261 minus 200-mesh.	2.2 on 10-mesh 25.2 minus 200-mesh.	75
M	1, 8-ft. by 36-in. ball mill.	Mill feed from fine ore bins passes to classifier, the sands from which are ground in ball mill in closed circuit with drag belt.	Minus 5-mm.	-	-
N	1, 6-ft. by 6-ft. ball mill.	Mill feed from fine ore bin to ball mill in closed circuit with classifier.	4.28 on 1/4-in., 5.95 minus 200-mesh.	-	72
O	3 No. "75" ball mills.	Mill feed to ball mills in closed circuit with classifier.	6.0 on 1-in., 7.3 minus 200-mesh.	17.2 on 48-mesh, 26.8 minus 200-mesh.	85
P	1, 6-ft. 6-in. by 12-ft. 1-in. rod mill.	Mill feed from fine ore bins to rod mill in closed circuit with classifier.	8.1 on 0.742-in., 5.0 minus 200-mesh.	1.0 on 20-mesh 19.8 minus 200-mesh.	-

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TABLE 20.- Operating data of ball and rod mills at various lead and zinc concentrators - Continued

Concentrator	Circulating load, percent of new feed	Grinding medium	Consumption of grinding medium per ton material ground, pounds	Mill liners, composition	Consumption of mill liners per ton material ground, pounds
A	Open circuit.	3-in. forged-steel balls.	2.2	Manganese steel.	-
B	Do.	Do.	2.0	Titanite steel.	1.0
C	-	Three brands of cast-iron balls in use.	Brand A-3.21 Brand B-3.25 Brand C-2.82	Electric steel and cast-iron.	-
H	-	3- and 4-in. cast-iron and steel balls.	2.19	Cast-iron.	.4
I	300	Steel.	1.986	Manganese steel.	.247
J	400	3-inch "Adamantine" steel balls.	2.7	Do.	.21
K	-	2.5- to 3.0-in. steel rods (C, 0.8 to 1.0%). 2.0- to 2.5-in. cast chilled balls.	Rods 1.2. Balls 1.1.	Do.	.21
L	340	5-in. chrome-steel balls.	3.24	Titanite	.71
M	-	3-in. cast balls.	3.19	-	-
N	300	4-in. chrome-steel.	1.5	Manganese steel.	-
O	400	4-1/2 in. forged-steel balls.	1.8	Manganese steel and chrome-molybdenum steel.	.3
P	200	2.5-in. manganese-chrome steel rods.	2.2	-	-

Vibrating screens are being used more and more in place of trommels. At the concentrators studied, classifying of the finer sizes of material is effected by desliming in cone or rake classifiers, and the material to be treated on tables is classified in hydraulic classifiers of the sorting-column type. Since the general adoption of flotation, much greater emphasis has been placed on preventing slime losses by proper desliming and thickening equipment.

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Pulp Densities and Fineness of Overflow

In most plants where grinding units are in closed circuit with rake-type classifiers, the classifier overflow generally will have a pulp density of about 25 to 40 percent solids. The amount of minus 200-mesh material in the overflow at different plants ranges from 30 to 80 percent of the total solids, depending upon the fineness of grinding necessary to liberate the mineral particles and to effect the desired grades and recoveries of the final products.

Gravity Concentration

Of the 16 plants studied, 6 employ gravity concentration supplemented by flotation. At 3 of these, both jigs and tables are used, at 2 jigs only, and at 1 tables only. Jig tailings are discarded at 3 of these plants, and at 2 the jig tailings are reground and treated by flotation.

The methods of jigging practiced at lead-zinc plants are: (1) Concentration of sized material and (2), concentration by roughing and cleaning. At concentrator C, the mill feed is crushed to 7 mm. A sized product between 7 mm and 0.13 inch is treated on one set of jigs, and the minus 0.13-inch plus 0.0325-inch material on another set. A lead product is obtained from the first two cells and the middlings and tailings are ground in ball mills for further treatment by flotation; fine sands are treated on tables. Grading the mill feed into a number of different sizes is practiced at concentrator D. Trommels are used to produce products of minus 30-mm, 18-mm, 12-mm, 7-mm, 3-mm, and 16-mesh sizes.

A roughing and cleaning system of jigging is practiced at concentrators A, B, and E. The mill feed is crushed to 5/8-inch or less and is deslimed and fed to 6- or 7-cell rougher jigs. The hutch products from the first 4 or 5 cells go to a cleaner jig, which produces final lead and zinc concentrates; and the hutch products from the last rougher cells, the rougher-bed middling, and cleaner-jig tailings are reduced in rolls and treated in middling or sand jigs by roughing and cleaning or by cleaning only.

In table concentration the dewatered fine material from various sources in the jig section of concentrator A is fed to a DeMier level jig. The hutch products from this jig go to a battery of 6 tables, while the bed middlings are ground in a ball mill and are returned to the jig. The tables produce a zinc concentrate, a zinc-lead product which is separated on another table, a zinc sand-middling which joins the jig-bed middlings in the ball mill, and tailings which are returned to the DeMier jig. The tailing from the jig is discarded. At concentrator C the classified fine sands are treated on tables. The lead product from the tables goes to lead bins, and the middlings and tailings are ground in ball mills and later treated by flotation. At other plants more elaborate systems of gravity concentration are employed. Tables are sometimes used incidentally in all-flotation plants as pilot machines to indicate the efficiency of flotation, the degree of oxidation, or the fineness of grinding obtained.

Flotation

Many different flow sheets and types of flotation equipment have been employed for concentration of lead and zinc ores. Although certain flotation reagents have become more or less standard for different classes of work, the quantities used varies considerably in different plants. Conditioning of the pulp before flotation requires careful study, not only to effect the best separation but also to determine the quantity of reagents necessary The flotation methods at the several plants studied may be classified as follows: (1)

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Treatment of fines collected in thickeners from classifier overflows resulting from dewatering gravity-concentration products; (2) treatment of all fine material not included in feed to jigs, and middlings and tailings from sand jigs; (3) all fines not included or recovered in gravity-concentration units, and reground middlings and tailings from jigs and tables; (4) entire mill feed ground and treated by flotation.

Table 21 gives the screen analyses of flotation feed from grinding units at 14 plants.

Flotation reagents used to effect differential flotation for producing lead and zinc concentrates at 8 plants are given in Table 22.

Grade of Products and Recoveries

The grade of lead and zinc concentrates produced and the recoveries obtained at various plants are given in table 23.

TABLE 21.- Screen analyses of flotation feed from grinding units

Concentrator	Screen sizes, mesh							
	35	48	65	100	150	200	-200	Total
								percent
A	.....	2.51	8.21	16.15	.....	35.37	37.76	100.00
C	.....	.3	2.2	8.4	11.3	12.2	65.6	100.00
D	.....	.3	2.5	5.5	.....	21.0	70.7	100.00
E	1.7	.....	11.7	14.4	10.0	13.4	48.8	100.00
F	.....	.....	.8	3.4	9.6	10.2	76.0	100.00
G	.....	.21	1.48	6.83	9.41	9.97	72.10	100.00
H	.....	.....	.2	2.3	6.0	10.1	81.4	100.00
I	.....	1.1	9.0	14.7	14.1	13.6	47.5	100.00
J	.....	.....	5.09	9.10	8.86	8.10	68.85	100.00
K	.4	1.3	3.7	9.1	4.2	21.2	60.1	100.00
L	.....	.....	1.37	8.22	17.80	26.02	73.98	100.00
N	.....	.....	.....	7.0	.....	18.0	75.0	100.00
O	.....	.....	5.13	4.23	8.66	6.11	75.87	100.00
P	.93	3.39	5.73	16.26	9.35	25.74	38.60	100.00

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TABLE 22.- Flotation reagents used to effect differential flotation for producing lead and zinc concentrates at various concentrators

Flotation reagents	Name of concentrator, designated by letter							
	D	G	H	I	J	N	O	P
	Reagent per ton treated, pounds							
Lead circuit -								
Soda ash	0.10		0.21	1.466	1.90	0.80		(2)
Sodium sulphite				.1818				
Sodium cyanide				.0616	.65	.40	0.060	0.0954
Zinc sulphate	.30	0.88	.20		.90	.50	1.395	
Pine oil						.07		
Cresylic acid	.10	.07			.50	.05	[1].175	.0307
Aerofloat	.03	.12						
Sodium xanthate				.1826	.10	.10		
Potassium xanthate				.0246			.161	.0239
Thiocarbonalid						.05		.1073
Barrett No. 4			.05					
Minerec "A"			.11					
Lime							.574	
Zinc circuit -								
Copper sulphate	.30	.61	.51	1.210	1.30	1.40	.783	2.2964
Lime				3.337	1.90	.20	1.848	.7956
Pine oil		.03		.125	.01		.151	.1840
Sodium xanthate		.04	.19		.10		.50	
Potassium xanthate				.0399				.0208
Barrett No. 4	.05	.06	.18					
Barrett No. 634						.05		
Aerofloat	.05							.1280
Grade of products[3] -								
Lead concts.								
Lead percent	61.3	70.8	75.2	61.06	59.9	58.37	37.81	53.05
Zinc percent	5.0	6.9	5.7	5.71	4.0	11.57	13.47	
Zinc concts.								
Lead percent	9.3	3.9	2.7	1.17	2.55	1.81	1.63	
Zinc percent	46.7	51.3	56.8	51.20	47.60	54.77	54.45	55.88

[1]Added to mica circuit ahead of lead circuit.
[2]Added when alkalinity falls too low.
[3]These are average grades over definite periods, whereas the use of reagents indicated may have been modified during these periods.

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TABLE 23.- Average grade of concentrates and recoveries at various concentrators treating lead and zinc ores

Concentrator	Heads assay, percent			Method of concentration	Lead concentrates						Zinc concentrate					
					Assay, percent			Recoveries, percent			Assay, percent			Recoveries, percent		
	Ag.oz	Pb	Zn		Ag.oz	Pb	Zn	Ag	Pb	Zn	Ag.oz	Pb	Zn	Ag	Pb	Zn
A	.....	.....	.....	Gravity 1/	.....	.....	.....	.....	.....	.....	.....	1.97	59.12	.....	.....	.....
				flotation.								1.90	59.72			
B	.....	1.7	6.35	Do. 1/	.....	74.0	5.2	.....	85.6	.....	.....	.36	61.50	.....	.....	58.3)84.3
												.07	59.80			26.0)
C	4.29	10.42	1.55	Do. 1/	24.11	58.19	3.13	89.70	89.12	32.15	5.90	3.77	47.37	1.05	0.28	23.29 2/
				Gravity 3/	27.09	52.02	3.5	83.8	85.4	63.4						
D	4.77	8.7	1.2	flotation,	34.6	61.3	5.0	89.5	93.2	33.3	10.1	9.3	46.7	4.7	2.6	55.5
				flot.jig tails	31.4	46.0	4.0	90.2	90.1	68.8						
E	.....	.....	2.9	Gravity, flotation.3/	.....	.....	.....	.....	.....	.....	.....	.....	60.19	.....	.....	90.15
F	.....	3.5	0.2	Gravity 4/	1.2	75.89	.5)	Total lead recovery between 96 and 97 percent; table con-								
				flotation	1.9	73.03	2.4)	centrates constitute about 52 percent of total output.								
G	4.32	10.95	2.79	Flotation.	27.2	70.8	6.9	88.9	91.5	35.1	5.3	3.9	51.3	3.7	1.0	55.0
H	3.4	9.0	6.5	Do.	24.9	74.2	5.8	81.1	91.0	9.7	4.8	2.8	56.2	14.2	3.1	85.7
I	9.09	6.21	5.01	Do.	50.19	61.06	5.71	53.61	95.46	11.08	30.86	1.17	51.2	26.81	1.48	80.76
J	9.44	5.95	4.14	Do.	87.10	59.9	4.0	65.4	71.3	6.8	14.28	2.55	47.6	9.5	2.7	72.4
L	13.0	11.36	1.65	Do.	32.05	33.43	2.12	77.26	92.14	40.19	.....	.....	.....	.....	.....	.....
M	8.17	5.13	5.98	Do.	74.02	57.05	10.06	75.49	92.59	14.02	12.4	2.79	52.36	11.92	4.23	68.29
N	1.155	1.99	9.93	Do.	25.22	58.37	11.57	62.95	81.86	.....	1.78	1.81	54.77	.....	.....	90.69
O	3.37	4.93	15.41	Do.	19.98	37.81	13.47	63.43	81.97	9.35	3.05	1.63	54.45	21.73	7.91	84.64
P	.....	1.5	11.6	Do.	.....	53.05	.....	.....	59.01	.....	.....	.....	55.88	.....	.....	81.42

1/ Jigs and tables.

2/ Based on mill feed.

3/ Jigs only.

4/ Tables only.

Power Consumption

The distribution of power consumed per ton of ore treated at various concentrators is presented in table 24.

Labor

The number of men required to operate a concentrator varies considerably. In general, a plant treating 5,000 tons per day will require a much smaller number of men proportionately than a plant treating only 500 tons. Table 25 indicates the labor distribution at seven concentrators of different capacities in terms of man-hours per ton and of tons treated per man, and Table 26 shows the over-all milling cost in terms of man-hours per ton at 116 plants treating lead and zinc ores.

Milling Costs

Milling costs per ton of ore treated at a number of plants are given in table 27. The method of keeping costs at different plants varies, so that it is usually difficult to present cost data covering several plants operated by different companies in a single table in which the separate cost items are in complete accord. Milling costs per pound of lead and zinc metals at various concentrators are given in table 28.

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TABLE 24.- Distribution of power consumed per ton of ore treated at various concentrators

Item	Concentrator B		Concentrator F		Concentrator I		Concentrator J		Concentrator K	
	Kw.hr.	Percent of total	Kw.hr.	Percent of total	Kw.hr.	Percent of total	Kw.hr.	Percent of total	Kw.hr.	Percent of total
Crushing	1.86	15.1	3.4	20.4	2.682	8.6	1.77	7.4	[1]1.51	5.57
Elevating	3.26	26.4								
Grinding	2.01	16.3	4.0	24.0	11.174	36.0	)		9.15	33.79
							)15.84	65.9		
Classification							)			
Screening										
Conveying										
Jigging	0.79	6.4								
Tabling	0.43	3.5	1.7	10.2					.25	.92
Flotation	1.22	9.9	4.6	27.5	11.514	37.0	4.85	20.2	8.24	30.40
Dewatering concts.					4.111	13.2	0.39	1.6	.59	2.18
Pumping	2.76	22.4							6.15	22.71
Water supply			2.0	12.0					.72	3.66
Tailings disposal			.9	5.3						
Miscellaneous			[2].1	.6	1.609	5.2	1.19	4.9	[3].48	1.77
Total	12.33	100.0	16.7	100.0	31.090	100.0	24.04	100.0	27.09	100.00

[1]Includes conveying.
[2]Loading of concentrates.
[3]Thickeners.

TABLE 24.- Distribution of power consumed per ton of ore treated at various concentrators - Continued

Item	Concentrator L		Concentrator M		Concentrator O		Concentrator P	
	Kw.hr.	Percent of total	Kw.hr.	Percent of total	Kw.hr.	Percent of total	Kw.hr.	Percent of total
Crushing	1.935	3.78	[4]6.05	21.33			[5]41.17	53.96
Elevating								
Grinding	19.180	37.48	)		[6]10.96	35.08	4.74	6.21
			) 9.95	34.94				
Classification	.214	.42	)				)	
Screening							) 8.20	10.74
Conveying	1.611	3.15					)	
Jigging								
Tabling					2.08	6.66		

[4]Includes screening.
[5]Crushers and rolls.
[6]Crushing not included as ore is crushed at mine to 1-1/2-inch size.

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TABLE 24.- Distribution of power consumed per ton of ore treated at various concentrators - Continued

Item	Concentrator L		Concentrator M		Concentrator O		Concentrator P	
	Kw.hr.	Percent of total	Kw.hr.	Percent of total	Kw.hr.	Percent of total	Kw.hr.	Percent of total
Flotation	16.050	31.36	7.94	27.96	11.73	37.53	19.61	25.70
Dewatering concts.	4.890	9.56	1.73	6.10	2.34	7.49	.....	.....
Pumping	5.760	11.25	.....	.....	.....	.....	.....	.....
Water supply	.....	.....	1.90	6.71	1.65	5.29	.....	.....
Tailings disposal	.319	.62	.85	2.96	.....	.....	.86	1.13
Miscellaneous	[7]1.221	2.38	.....	.....	[8]2.48	7.95	.....	.....
Total	51.170	100.00	28.42	100.00	31.24	100.00	76.30	100.00

[7]Control and mill sampling 0.712 kw. hr.
[8]Thickening, sampling, lighting, and laboratory.

TABLE 25.- Distribution of labor, man-hour per ton treated and tons per 8-hour shift

Item	Concentrator B 40 tons per hour			Concentrator F 208 tons per hour		
	Man-hour per ton	Tons per 8-hour shift	Percent of total	Man-hour per ton	Tons per 8-hour shift	Percent of total
Sorting	(1)	(1)	.....	.....	.....	.....
Crushing	0.0374	214	9.1	0.0235	340	22.0
Grinding	.0187	428	4.5	.0141	567	13.25
Classification, screening, and conveying	.0187	428	4.5	.....	.....	.....
Concentration: Gravity	[2].1496	[2]53.5	36.4	[3].0141	[3]567	13.25
Flotation	.0748	107	18.2	.0094	850	8.8
Dewatering concentrates	.0374	214	9.1	[4].0047	[4]1,700	4.4
Weighing and loading	.....	.....	.....	.....	.....	.....
Sampling	.....	.....	.....	.0047	1,700	4.4
Tailings disposal	.0374	214	9.1	.....	.....	.....
Maintenance	.....	.....	.....	.0283	283	26.6
Supervision	.....	.....	.....	.0078	1,020	7.3
Assaying	.....	.....	.....	.....	.....	.....
Warehouse	.....	.....	.....	.....	.....	.....
Power	.0374	214	9.1	.....	.....	.....
Miscellaneous	.....	.....	.....	.....	.....	.....
Total	.4114	19.45	100.0	.1066	75	100.00

[1]Repairs included in the different items.
[2]Jigs and tables.
[3]Table concentration.
[4]Includes loading.

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TABLE 25.- Distribution of labor, man-hour per ton treated and tons per 8-hour shift - Continued

Item	Concentrator H 50 tons per hour			Concentrator I 17 tons per hour		
	Man-hour per ton	Tons per 8-hour shift	Percent of total	Man-hour per ton	Tons per 8-hour shift	Percent of total
Sorting						
Crushing	0.114	70	21.4	0.1015	78.79	16.5
Grinding	[5].114	[5]70	21.4	.0768	104.14	12.5
Classification, screening, and conveying						
Concentration· Gravity						
Flotation	.176	45.5	33.0	.0768	104.14	12.5
Dewatering concentrates	.030	267	5.6	.0768	104.14	12.5
Weighing and loading				.0512	156.21	8.35
Sampling				[6].0256	[6]312.43	4.2
Tailings disposal				.0512	156.21	8.35
Maintenance				.0512	156.21	8.35
Supervision				.0256	312.43	4.2
Assaying				.0512	156.21	8.35
Warehouse				.0256	312.43	4.2
Power						
Miscellaneous	.099	81	18.6			
Total	.533	15	100.0	.6135	13	100.00

[5]Includes classifying.
[6]Includes mixing reagents.

TABLE 25.- Distribution of labor, man-hour per ton treated and tons per 8-hour shift - Continued

Item	Concentrator J 13 tons per hour			Concentrator L 7 tons per hour			Concentrator P 20 to 25 tons per hour		
	Man-hour per ton	Tons per 8-hour shift	Percent of total	Man-hour per ton	Tons per 8-hour shift	Percent of total	Man-hour per ton	Tons per 8-hour shift	Percent of total
Sorting	0.027	296	8.0						
Crushing	.047	170	13.9	0.1040	77.0	11.5	0.0955	83.8	31.4
Grinding	.078	102.6	23.1	.0995	80.4	11.0	.0579	138.2	19.0
Classification, screening, and conveying							.0222	360	7.3
Concentration Gravity									
Flotation	.097	82.5	28.7	.3125	25.6	34.5	.1067	75	35.0

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TABLE 25.- Distribution of labor, man-hour per ton treated and tons per 8-hour shift - Continued

Item	Concentrator J 13 tons per hour			Concentrator L 7 tons per hour			Concentrator P 20 to 25 tons per hour		
	Man-hour per ton	Tons per 8-hour shift	Percent of total	Man-hour per ton	Tons per 8-hour shift	Percent of total	Man-hour per ton	Tons per 8-hour shift	Percent of total
Dewatering concentrates	[6]0.022	[6]364	6.5	0.0994	80.5	11.0			
Weighing and loading									
Sampling				.0957	83.6	10.6			
Tailings disposal				.0762	104.9	8.4	0.0222	360	7.3
Maintenance									
Supervision									
Assaying									
Warehouse									
Power									
Miscellaneous	.067	119.4	19.8	.1180	67.7	13.0			
Total	.338	23.7	100.0	.9053	8.83	100.0	.3045	26.3	100.0

[6]Includes mixing reagents.

TABLE 26.- Over-all milling costs in terms of man-hour per ton and tons per man-shift

Number of mills reporting	Total tons milled	Principal metal produced	Average tons per man-shift	Average man-hour per ton
14	7,359,063	Lead	33.170	0.2412
19	2,630,082	Zinc	[1]13.135	.6091
83	7,957,868	Zinc and lead	[1]21.437	.3739

[1]Some mills reported 10-hour mill shifts; figures from these mills have been adjusted to an 8-hour-shift basis.

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TABLE 27.- Milling costs per ton of ore treated at various concentrators

Item	Concentrator B Period: Aug., Sept., Oct. 1929				Concentrator C Period: Year 1929				Concentrator F Period: Year 1929
	Labor	Supplies and misc.	Power	Total	Labor	Supplies and misc.	Power	Total	Total costs
Sorting									
Crushing	$0.014	$0.049	$0.023	$0.086	$0.035	$0.034	$0.027	[1]$0.096	$0.0671
Grinding		.054	.024	.078	.019	.109	.045	.173	.0532
Elevating and conveying	.014	.012	.039	.065					[2].0119
Screening and classifying					.019	.023	.006	[3].048	
Jigging	.052	.002	.010	.064	.055	.042	.041	.138	.0634
Tabling	.039	.015	.005	.059	.010	.010	.011	.031	
Flotation	.036	.039	.015	.090	.066	.135	.032	.233	.0773
Dewatering concentrates									[4].0288
Disposal of tailings									.0045
Water supply		.004	.033	.037					.0306
Pumping									[5].0044
Power									
Sampling									
Assaying		.018		.018					
Maintenance									
Supervision									
Miscellaneous		.007		.007					
Overhead								.359	
Total	.155	.200	.149	.504	.204	.353	.162	1.078	.3412

[1]Includes conveying.
[2]Tramming crushed ore to concentrator.
[3]Screening only.
[4]Includes drying.
[5]Pumping slimes to flotation plant.

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TABLE 27.- Milling costs per ton of ore treated at various concentrators - Continued

Item	Concentrator G Period: Year 1930				Concentrator H Summarized ave.	Concentrator J Period: March 1930			
	Labor	Supplies and misc	Power	Total	Total costs	Labor	Supplies and misc	Power	Total
Sorting						$0.017	$0.001		$0.018
Crushing	$0.037		$0.017	$0.054	$0.060	.035	.021	$0.023	.079
Grinding	.070	$0.210	.131	.411	.130	.054	.181	.210	[6].445
Elevating and conveying					.020				
Screening and classifying					.020				
Jigging									
Tabling									
Flotation	.058	.120	.114	.292	.242	.073	.365	.064	.502
Dewatering concentrates	.018		.013	.031	[7].020	.015	.001	.005	.021
Disposal of tailings									
Water supply	.009		.007	.016	.010				
Pumping					.020				
Power					.270				
Sampling	.003		.002	.005					
Assaying									
Maintenance					[8].090				
Supervision	.043			.043					
Miscellaneous	.023	.030		[9].053	[10].130	.050	.151	.016	[11].217
Overhead									
Total	.261	.360	.284	.905	1.012	.244	.720	.318	1.282

[6] Includes classifying.

[7] Filters and thickeners.

[8] Plant maintenance and repair shops.

[9] Includes $0.013 for heating.

[10] Includes $0.01 for launders, $0.02 for transmission, $0.01 for lubrication, $0.02 for loading, $0.04 for lighting and heating, and $0.01 for change room.

[11] General mill expense.

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TABLE 27.- Milling costs per ton of ore treated at various concentrators - Continued

Item	Concentrator L Period: Year 1929				Concentrator M Period: Oct. 1, 1929-Apr. 1, 1930				Concentrator N Period: Sept., Oct. 1929
	Labor	Supplies and misc.	Power	Total	Labor	Supplies and misc.	Power	Total	Total costs
Sorting									$0.0472
Crushing	$0.0854	$0.0183	$0.0195	$0.1232	$0.1493	$0.0954	$0.0765	[12]$0.3212	.1110
Grinding	.0778	.1830	.1938	.4546	.0233	.1029	.1258	[13].2520	.3553
Elevating and conveying									
Screening and classifying	.0053		.0022	.0075					
Jigging									
Tabling									
Flotation	.2299	1.2652	.1621	1.6572	.1347	.3631	.1003	.5981	.5262
Dewatering concentrates	.0993	.0195	.0494	.1682	.0204	.0080	.0219	.0503	[14].0817
Disposal of tailings	.0535	.0054	.0032	.0621	.0408	.0110	.0105	.0623	
Water supply		.2540		.2540	.0506	.0008	.0241	.0755	
Pumping									
Power									
Sampling	.0742	.0143	.0072	.0957					
Assaying		.1710		.1710					.1114
Maintenance									
Supervision									
Miscellaneous	.1049	.0389	.0796	.2234	.1675	.6751		[15].8426	.0731
Overhead									[16].2447
Total	.7303	1.9696	.5170	3.2169	.5866	1.2563	.3591	2.2020	1.5506

[12]Includes screening.
[13]Includes classifying.
[14]Filters and thickeners.
[15]Includes assaying, overhead, shop, and operation of pumping plant and pipe line from river when used.
[16]General expense.

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TABLE 27.- Milling costs per ton of ore treated at various concentrators - Continued

Item	Concentrator O Period: Year 1930			
	Labor	Supplies and misc	Power	Total
Sorting				
Crushing				
Grinding	$0.0500	$0.1546	$0.0670	$0.2716
Elevating and conveying				
Screening and classifying				
Jigging				
Tabling	.0226	.0028	.0128	.0382
Flotation	.1157	.3320	.0723	.5200
Dewatering concentrates	.0202	.0035	.0161	[17].0398
Disposal of tailings	.0002	.0006		.0008
Water supply	.0003	.0432	.0102	.0537
Pumping				
Power				
Sampling	.0124	.0022	.0059	.0205
Assaying		.0595		.0595
Maintenance				
Supervision	.0888	.0007		.0895
Miscellaneous	.0201	.0468		[18].0669
Overhead		.0293		[19].0293
Total	.3303	.6752	.1843	1.1898

[17]Includes thickening.

[18]Includes $0.0071 for heating, $0.018 for mill clean-up, $0.032 for experimental work, $0.004 for lubrication, and $0.0045 for lighting.

[19]Royalties.

TABLE 28.-Milling costs per pound of combined lead and zinc metals at various concentrators

Concentrator	Period covered	Tons of ore milled	Tons of lead concentrates produced	Tons of zinc concentrates produced	Pounds of lead in lead concentrates	Pounds of zinc in zinc concentrates	Pounds of combined lead and zinc
B	Aug., Sept., Oct. 1929	24,974	49	2,192	72,520	2,673,176	2,745,696
C	Year 1929	451,111	72,002.56	3,448.19	83,796,579	3,266,815	87,063,394
C	Year 1931[1]	460,366	85,542	4,296	88,253,071	4,642,383	92,895,454
E	Year 1929	624,759		[2]27,136		32,666,740	32,666,740
G	Year 1930	97,447	12,244	2,073	16,676,328	2,139,336	18,715,664

[1]Bunker Hill & Sullivan Mining & Concentrating Co., Forty-Fourth Annual Report, for Year Ended Dec. 31, 1931.

[2]Calculated.

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TABLE 28.- Milling costs per pound of combined lead and zinc metals at various concentrators - Continued

Concentrator	Period covered	Tons of ore milled	Tons of lead concentrates produced	Tons of zinc concentrates produced	Pounds of lead in lead concentrates	Pounds of zinc in zinc concentrates	Pounds of combined lead and zinc
H	Year 1930	361,372	39,833	36,008	59,112,172	40,472,992	99,585,164
J	March 1930	9,930	703.5	625	842,793	595,000	1,437,793
L	Year 1929	37,834	11,852.7	.....	7,924,715	.....	7,924,715
M	Oct. 1, 1929 to Apr. 1, 1930	41,246	3,435	3,215	3,919,335	3,366,748	7,286,083
N	Sept., Oct. 1929	9,222	255.2	1,465.8	297,920	1,605,637	1,903,557
O	Year 1930	151,943	[2]16,244	[2]36,406.6	12,283,715	39,645,584	51,929,299

[2]Calculated.

TABLE 28.- Milling costs per pound of combined lead and zinc metals at various concentrators - Continued

Concentrator	Total cost of milling	Cost per pound of combined lead & zinc	Remarks
B	$12,586.90	$0.00458	No precious metals.
C	486,297.66	.00558	Lead concentrates contained 24.11 oz. Ag per ton and zinc concentrates 5.9 oz. Ag per ton, a total of 1,756,671 oz. Ag in combined concentrates.
C	384,098.47	.00414	Lead concentrates contained 19.95 oz. Ag per ton and zinc concentrates 4.36 oz. Ag per ton, a total of 1,726,293 oz. Ag in combined concentrates.
E	213,167.77	.00653	No precious metals.
G	88,189.54	.00471	Lead concentrates contained 30.7 oz. Ag per ton and zinc concentrates 5.6 oz. Ag per ton, a total of 387,500 oz. Ag in combined concentrates.
H	371,851.79	.00373	Lead concentrates contained 24.9 oz. Ag per ton and zinc concentrates 4.8 oz. Ag per ton, a total of 1,164,680 oz. in combined concentrates.
J	12,730.26	.00886	Lead concentrates contained 87.1 oz. Ag per ton and zinc concentrates 14.28 oz. Ag per ton, a total of 70,200 oz. in combined concentrates.
L	121,708.19	.01536	Lead concentrates contained 0.255 oz. Au and 32.C5 oz. Ag per ton, a total of 2666.86 oz. Au and 379,879 cz. Ag.
M	90,823.69	.01246	Lead concentrates contained 0.674 oz. Au and 74.02 oz. Ag per ton and zinc concentrates 0.07 oz. Au and 12.4 oz. Ag a total of 2,5C4.25 oz. Au and 294,125 oz. Ag.

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TABLE 28.- Milling costs per pound of combined lead and zinc metals at various concentrators - Continued

Concentrator	Total cost of milling	Cost per pound of combined lead & zinc	Remarks
N	14,299.63	.00752	Lead concentrates contained 25.22 oz. Ag per ton and 3.68 percent Cu, and zinc concentrates 1.78 oz. Ag per ton and 1.55 percent Cu, a total of 9,045 oz. Ag and 64,223 lb. Cu.
O	180,781.82	.00348	Lead concentrates contained 0.768 oz. Au, 19.98 oz. Ag per ton, and 4.02 percent Cu, and zinc concentrates 0.036 oz. Au and 3.05 oz. Ag, per ton and 1.0 percent Cu, a total of 13,783.8 oz. Au, 435,693 oz. Ag, and 2,034,004 lb. Cu.

Costs of Mining and Milling

Costs of the various separate operations connected with the mining and milling of lead and zinc ores have been presented in the preceding pages, together with summaries of mining and of milling costs at a number of properties. In only a few instances were dollar costs of both mining and milling available for the same property.

However, data were available at a large number of properties for the year 1930, from which could be calculated the man-hours per ton and tons per man-shift for both mining and milling These data are summarized in table 29, the figures being weighted averages in each instance. Labor usually accounts for about 60 percent of the total mining cost and for 20 to 40 percent of the total milling cost.

Except for the data in table 28, the costs presented in this paper have been costs per ton of ore mined or milled.

Since the value of an ore is based upon the quantity and value of the metal recoverable therefrom, costs per unit of metal for the various operations involved in its production would have greater significance than costs per ton of ore. Unfortunately, data upon which costs can be computed on this basis are very limited, since few companies publish costs in sufficient detail for this purpose. Table 30 presents operating costs per pound of lead and zinc contained in concentrates and (or) direct shipping ore, for mining, milling, general expense, plant depreciation, and taxes. These have been computed from published annual reports of the companies. For the sake of simplicity and uniformity, the costs are based on only the lead contained in lead concentrates or the zinc in zinc concentrates, disregarding lead in zinc concentrates and zinc in lead concentrates. Also, no account is taken of the value of precious metals, which in some instances may provide a large revenue and, if credited against the cost of the base metals, would reduce the tabulated costs materially.

TABLE 29.- Costs of mining and milling in terms of tons per man-shift and man-hours per ton

Number of mines and mills reporting	Total tons	Principal metal	Tons per man-shift				Man-hours per ton			
			Underground	Surface	Mill	Total	Underground	Surface	Mill	Total
13	7,211,115	Lead	9.030	75.439	33.314	6.494	0.8859	0.1060	0.2401	1.232
19	2,630,082	Zinc	5.1396	29.149	[1]13.135	3.278	1.5564	.2744	.6091	2.440
83	6,547,219	Lead and zinc	4.524	21.464	20.595	3.162	1.7683	.3728	.3884	2.5295

[1]Some mines reported 10-hour mill shifts; figures from these mines have been adjusted to an 8-hour shift basis.

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TABLE 30.- Operating costs per pound lead and zinc (not including smelting)

Company	Year	Lead in lead concentrates pounds	Zinc in zinc concentrates pounds	Combined lead and zinc pounds	Cost per pound combined lead and zinc: Mining and development	Milling	Mining and milling	General expense, local taxes, etc.	Depreciation and federal taxes	Total
Bunker Hill and Sullivan	1931	88,253 071	4,642 323	1/92,895,454	$0.0171	$0.0041	$0.0212	Included in mining and milling.	.....	$0.0212
Hecla	1931	47.893.419	758,322	2/48 651.741	.0132	3/.0025	.0157	$0.0041	.....	.0198
Tintic Standard	1931	4/27,986,298	.....	27,986,298	0214	.0033	.0247	.....	$0.0019	.0266
Silver King	1931	25 685,406	12,292,038	5/38.017,974	.....	.....	.0301	.0019	0009	0329
Coalition		6/40.530								
		25,725,936								
Park-Utah	1930	28.040.599	34 073,606	7/62,114,205	.0213	8/.0017	.0230	.0020	.0017	0267
Treadwell-Yukon	1931	8,287 978	5.994,946	9/14,282.924	.....	.....	.....	.....	.....	.0230
Tibo, Nev										
Tri-State district 10/	1930	11/72,787,200	12/473,350,800	546,138.000	.....	.....	.0189	13/.0037	14/.0043	.0269

1/ 209.400 lb. Pb in Zn concentrates. 5,284,804 lb. zinc, the Pb concentrates not included; 1,725.733 oz. Ag produced also.

2/ Pb in ore instead of in concentrates. Ore contained 2,995.39 oz. Au, 1,991,745 oz. Ag, and 424,754 lb. Cu.

3/ Pb in lease ore

4/ 390,924 lb. lead in Zn concentrates, 793,010 lb. Cu, 2,203 47 oz. Au. and 1,531.236 oz Ag not included.

5/ Milling and ore expense.

6/ Total Pb and Zn content; contained also 804,957 lb. Cu, 17,679 oz. Au, and 1,598,983 oz. Ag.

7/ Includes metal in direct smelting ore. Concentrates and smelting ore contained 1,271,869 oz. Ag.

8/ Milling and ore haulage

9/ Concentrates contained 922.59 oz. Au and 632,177 oz. Ag.

10/Preliminary figures, Tri-State Lead and Zinc Producers Association.

11/Estimated from preliminary figures of Tri-State Lead and Zinc Producers Association, assuming 80 percent lead in lead concentrates.

12/Estimated on assumption zinc concentrates average 60 percent Zn.

13/Includes taxes excluding income tax, insurance and royalty of $0 0026 per pound.

14/Depreciation and depletion.

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